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A Study of Waste Generation, Treatment and Disposal in the Metals Mining Industry

Midwest Research Inst, Kansas City, Mo

Prepared for

Environmental Protection Agency, Washington, D C Office of Solid Waste Management Programs

Oct 76

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A STUDY OF WASTE GENERATION, TREATMENT AND DISPOSAL IN THE METALS MINING INDUSTRY

This final report (SW-132c) describes work performed
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16. Abstracts

The primary objective of the program was to provide EPA with detailed information concerning the generation, treatment, and disposal of potentially hazardous wastes in the metals mining and concentrating industries. The definition of potentially hazardous wastes used in this study is as follows:

> "Any land disposed wastes that contain one or more hazardous substances in concentrations above that of the land in or on which it is disposed are considered potentially hazardous."

The metals mining and concentrating industries covered in this study were categorized by the following Bureau of the Census Standard Industrial Classification Numbers: 1021 - Copper Ores; 1031 - Lead-Zinc Ores; 1092 - Mercury Ores; 1094 - Uranium, Radium and Vanadium Ores; and 1099 - Metal Ores--not elsewhere classified - antimony, beryllium, platinum, rare earths, tin, titanium, and zirconium.

Potentially hazardous wastes result from waste rock, overburden, and concentrator tailings. All potentially hazardous land-disposed wastes generated by this industry are disposed on the property owned by industry companies. No outside contractors are used and no off-site disposal is practiced.

Waste disposal and treatment practices are discussed, and estimates are given for the cost of hazardous waste treatment and disposal at typical facilities.

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PREFACE

This report presents the results of a study of waste generation, treatment, and disposal in five categories of the metals mining industry. The categories are SIC (Standard Industrial Classification) Nos. 1021 (Copper Ores), 1031 (Lead and Zinc Ores), 1092 (Mercury Ores), 1094 (Uranium and Vanadium Ores), and 1099 (Metal Ores, Not Elsewhere Classified).

The study was conducted by Midwest Research Institute (MRI) for the U.S. Environmental Protection Agency (EPA) under Contract No. 68-01-2665. The EPA project officer was Allen Pearce of the Hazardous Waste Management Division, Office of Solid Waste Management Programs.

The principal authors of this report and their areas of responsibility are David Bendersky, Project Leader; Robert E. Gustafson, Industry Characterization; Charles E. Mumma, and Kenneth R. Walker, Waste Generation and Characterization; E. Patrick Shea, Waste Treatment and Disposal Techniques; and Dennis Costello, Cost Analysis. Other MRI contributors are Patricia Levy, John L. Sealock, Jr., and Alan Woodall. Carl Christiansen, University of Missouri School of Mines and Metallurgy, and Karl C. Dean, formerly with U.S. Bureau of Mines, served as project consultants.

Many other individuals and organizations contributed to this study. The following were especially helpful: Sam Morekas and Timothy Fields, Jr., U.S. EPA, Office of Solid Waste Management Programs, Hazardous Waste Management Division, for their guidance throughout the project; Brice O'Brien, American Mining Congress, and Edward C. Bingham, Copper Range Company, for their assistance in arranging contacts with the mining industry. Various Federal and State Government agencies provided important data, as referenced in the report. Finally, the cooperation of the mining companies and their representatives who provided pertinent information on their operations during our site visits and via AMC questionnaires is gratefully acknowledged.

Approved for:

MIDWEST RESEARCH INSTITUTE

L. J. Shannon

L. J. Shannon, Director
Environmental and Materials Sciences Division

October 22, 1976

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EXECUTIVE SUMMARY

This report is the result of a study commissioned by the U.S. Environmental Protection Agency to assess the waste generation, treatment, and disposal practices in the metals mining industry. This study is one of a series of industry studies by the Office of Solid Waste Management Programs, Hazardous Waste Management Division. The studies were conducted for information purposes only and not in response to a Congressional regulatory mandate. As such, the studies serve to provide EPA with: (1) an initial data base concerning the current and projected types and quantities of industrial wastes, applicable treatment and disposal technologies and their associated costs; (2) a data base for technical assistance activities; (3) a background for guidelines development work pursuant to Section 209 of the Solid Waste Disposal Act as amended.

The definition of "potentially hazardous waste" in this study was developed based upon contractor investigations and professional judgment. This definition does not necessarily reflect EPA thinking since such a definition, especially in a regulatory context, must be broadly applicable to widely differing types of waste streams. The presence of a toxic, flammable, explosive or reactive substance should not be the major determinant of hazardousness if there are data to represent or illustrate actual effects of wastes containing these substances in specific environments. Thus, the reader is cautioned that the data presented in this report constitute only the contractor's assessment of the hazardous waste management problems in this industry. EPA reserves its judgments pending a specific legislative mandate.

The primary objective of this program was to provide EPA with detailed information concerning the generation, treatment, and disposal of potentially hazardous wastes in the metals mining and concentrating industries. The metals mining and concentrating industries covered in this study were categorized by the following Bureau of the Census Standard Industrial Classifications Numbers: 1021 - Copper Ores; 1031 - Lead-Zinc and Zinc Ores; 1092 - Mercury Ores; 1094 - Uranium, Radium, and Vanadium Ores; and 1099 - Metal Ores--not elsewhere classifed - antimony, beryllium, platinum, rare earths, tin, titanium, and zirconium. The contract specified that particular attention be given to those wastes which may contain the following materials: asbestos; arsenic; lead; mercury; halogenated hydrocarbons; pesticides; selenium; and zinc. Other substances which were believed to be potentially hazardous were also included, such as carcinogens, chromium, and cadmium.

The definition of potentially hazardous wastes used in this study is as follows:

> "Any land disposed wastes that contain one or more hazardous substances in concentrations above that of the land in or on which it is disposed are considered potentially hazardous."

The approach followed in this project was: first, define the industry and collect and report statistical and historical data on production of ore and waste from mines and production of metal concentrates and wastes from concentrators (utilizing secondary sources); second, contact the American Mining Congress for assistance in obtaining information from the mine and concentrator operators; third, contact the State Department of Mineral Resources in each state where the ores of interest were mined and concentrated; fourth, contact other mining associations, notably the Lead and Zinc Association and the American Quicksilver Institute; and finally, visit selected mines and concentrators to obtain all available data on the operation, the quantities and composition of the various waste materials, and the methods used for land disposal of the wastes.

The open literature was searched, and the Bureau of Mines, the EPA regional laboratories and offices, and the U.S. Geological Survey offices and laboratories were contacted for information on the activities in the mining and concentrating industries of concern. The U.S. Geological Survey was contacted for information on the background concentrations of minerals in and around the mines. A questionnaire was circulated by the American Mining Congress to 86 mining companies; 45 responses were received covering 64 mines and 52 concentrators. The Lead and Zinc Association was asked to assist in contacting mining companies for permission to visit their operations. The American Quicksilver Institute was asked to assist in contacting mercury mining and concentrating companies for permission to visit their operations. The relevant state mineral departments were solicited for information on mining activities in their states; 10 replied.

All of this information was analyzed, and representatives of mining companies in the five SIC categories were contacted for permission to visit their facilities. One company refused to grant permission for a visit and also failed to return the AMC questionnaire.

There were 27 site visits covering 47 mines and 40 concentrators out of a total of 274 mines and 138 concentrators in the United States. Additionally, we received questionnaires from 15 companies covering 39 mines and 31 concentrators we had not visited. In all we gathered first hand information on 86 out of 274 mines, and 71 out of 138 concentrators.

The metals mining industry mines and processes tremendous tonnages of ore which vary considerably in composition from mine to mine and even within a single mine. Furthermore, the industry is "easy in - easy out," i.e., quick to shut down or reopen existing mines in response to changes in profitability and demand. Statistics in recent years thus bear a definite element of uncertainty. Projections for future years bear a considerable element of uncertainty. The statistics in the report generally are reported to more significant figures than are warranted, mainly to facilitate the tasks of making percentages add to 100, adding columns and rows to consistent totals, and maintaining consistency from table to table. The reader is advised to mentally round off the data as he uses them and to use and interpret them as representative statistics rather than as precise or exact data.

Industry Characterization

Metals mining is one of the oldest industries in the United States, with early activities dating back to the 1600's. Large-scale operations were begun over 100 years ago. The modern copper mining industry dates from about 1840; mercury mining started about 1850; and large-scale lead and zinc mining operations began in the 1860's. A latecomer is uranium mining, which became prominent with the introduction of the "atomic age" in the 1940's.

Most of the five metal mining industries covered in this study, with the exception of mercury, are still very active. The copper ore, lead-zinc and zinc ores, mercury ore, uranium-vanadium ores, and miscellaneous ores industries had a total of 274 mines and 138 concentrators in operation at the end of 1974. The breakdown for each industry is given in Table 1. The uranium-radium-vanadium industry had the largest number of active mines, followed by copper, lead-zinc, miscellaneous ores, and mercury.

TABLE 1

ACTIVE MINING AND CONCENTRATING OPERATIONS FOR 1974

SIC	Ores	Mines	Concentrators
1021	Copper	57	61
1031	(a) Lead-zinc	39	40
	(b) Zinc	9	7
1092	Mercury	2	2
1094	Uranium-vanadium	158	17
1099	Miscellaneous ores	9	11
Total		274	138

Copper ore production is concentrated in six states (Arizona, Utah, Montana, New Mexico, Nevada, and Michigan), which produced 99 percent (381 million metric tons (419 million tons)) of the nation's output in 1973. Domestic lead-zinc and zinc ore production is located in 15 states, with over 45 percent (8 million metric tons (8.8 million tons)) in Missouri. Uranium mines are located in six states, with New Mexico and Wyoming providing more than 75 percent (4 million metric tons (4.4 million tons)) of the output. Vanadium is presently mined only in Arkansas, and mercury is mined only in California. Miscellaneous metals (antimony, beryllium, platinum, titanium, zirconium, and rare earths) are mined in small quantities in eight states, with the largest tonnage in Florida.

The number of returned questionnaires and mines and concentrators visited (no duplicates) are shown in Table 2.

Composite statistics in 1974 for the five metals mining industries covered in this study are shown in Table 3. A total of 430 million metric tons (473 million tons) of raw ore was mined by the five industries, from which 15 million metric tons (16.5 million tons) of concentrated ore and miscellaneous concentrates were produced.

The data collected in the site visits and furnished in the questionnaires were supplemented with data from the U.S. Bureau of Mines Mineral Specialists and the Engineering & Mining Journal to arrive at the composite statistics for this industry. Where necessary to complete missing data, engineering estimates were used. It was assumed that the ore produced, waste generated, and concentrates produced by the mines and concentrators for which data could not be found was the average of the values reported to the investigators.

Waste Generation

In the process of mining and concentrating the ores, 769 million metric tons (847 million tons) of dry waste, and 1,380 million metric tons (1,518 million tons) of wet waste* were generated in 1974. The mining wastes generated consisted of 371 million metric tons (409 million tons) of waste rock and 139 million metric tons (153 million tons) of overburden. There were 259 million metric tons (285 million tons) of dry concentrator wastes and 869 million metric tons (956 million tons) of wet waste.

Region IX was by far the leading generator of metals mining and concentrating wastes, with 55 percent (427 million metric tons) of the national total of dry wastes. Region VIII ranked second with 25 percent (193 million metric tons), and Region VI was third with 15 percent (119 million metric tons). These three regions produced 95 percent (738 million metric tons) of the national metals mining and concentrating industry wastes in 1974.

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*Wet waste = dry waste + water

TABLE 2

QUESTIONNAIRES RECEIVED OR MINES AND CONCENTRATORS VISITED

Ore type	Mine type Under-ground	 Open-pit	 In situ	% of Mines	Concentrator	% of Concentrators
Copper	13	15	1	51	30	49
Lead-zinc	23	-	-	60	18	40
Zinc	7	-	-	78	5	71
Uranium-radium	8	11	-	12	10	59
Vanadium	-	3	-	100	1	100
Miscellaneous	1	4	-	55	7	64
Total	52	33	1		71	

TABLE 3

COMPOSITE STATISTICS FOR METALS MINING INDUSTRIES SICs 1021, 1031, 1092, 1094, AND 1099 FOR 1974 (METRIC)

			Products (10^3 TPY)				Wastes (10^3 TPY)					Ratio of dry		% of Total Waste by region	
State	Region	Ore mined (10^3 TPY)	Primary	Other metals	Miscellaneous	Total	Waste rock	Overburden	Concentrator tailings (dry)	Concentrator tailings (wet)	Total waste	Waste to ore	Waste to product	Dry	Wet
Maine	I	48	4	3	0	7	1	0	40	160	161	0.85	5.9	0.01	0.02
New Jersey		7,183	172	0	0	172	0	0	125	627	627	0.63	0.73		
New York		1,306	227	146	0	373	5	0	934	4,670	4,675	0.72	2.52		
	II	8,489	399	146	0	545	5	0	1,059	5,297	5,302	0.71	1.95	0.14	0.61
Pennsylvania		544	54	0	41	95	0	0	406	2,265	2,265	0.75	4.27		
Virginia		544	28	4	404	436	12	0	141	2,091	2,103	0.28	0.35		
	III	1,088	72	4	445	521	12	0	547	4,356	4,368	0.51	1.07	0.07	0.50
Florida		15,422	388	125	42	555	0	0	34	374	374	0.002	0.06		
Kentucky		59	2	2	54	58	0	0	0	0	0	-	-		
Tennessee		5,180	229	0	3,796	4,025	101	0	781	17,395	17,496	0.17	0.22		
	IV	20,661	619	127	3,892	4,638	101	0	815	17,769	17,870	0.04	0.20	0.12	2.04
Illinois		294	58	61	73	192	5	0	102	510	515	0.4	0.56		
Michigan		8,059	246	0	0	246	7,836	0	7,292	34,618	42,454	1.9	61.5		
Wisconsin		468	15	12	0	27	0	0	441	2,205	2,205	0.99	16.3		
	V	8,821	319	73	73	465	7,841	0	7,835	37,333	45,174	1.8	33.7	2.04	4.29
Arkansas		363	5	0	0	5	45	962	368	1,587	2,494	3.5	255		
New Mexico		22,505	628	65	0	693	54,997	23,965	21,807	70,758	147,562	4.5	145		
Oklahoma		213	4	0	0	4	8,165	181	209	836	9,182	40.2	2,140		
Texas		367	1	0	0	1	484	7,263	367	2,276	10,023	22	8,114		
	VI	23,448	638	65	0	703	63,691	32,271	22,751	75,457	169,261	5.06	169	15.44	8.68
Missouri	VII	8,174	639	196	30	865	1,338	0	7,309	31,079	32,417	1.06	10	1.12	3.57
Colorado		1,506	71	44	0	115	37	0	1,394	9,814	9,851	0.95	12.4		
Montana		56,471	366	< 0.1	< 0.1	366	9,077	0	56,099	196,433	205,510	1.15	178		
Utah		135,921	895	24	341	1,260	15,526	513	35,645	114,757	130,796	0.37	40.4		
Wyoming		1,860	4	0	0	4	353	72,224	1,860	5,889	78,466	40	18,609		
	VIII	195,758	1,336	68	341	1,745	24,983	72,737	94,998	326,893	424,623	0.98	110	24.97	37.61
Arizona		148,417	4,877	56	0	4,933	235,868	30,393	109,904	320,274	586,535	2.54	76.3		
California*		22	21	14	2	37	908	487	166	986	2,381	108	3.9		
Nevada		12,294	358	-	0	358	36,287	1,963	12,156	37,156	25,406	4.10	141		
	IX	160,733	5,256	70	2	5,328	273,063	32,843	122,226	358,416	664,322	2.66	80.1	55.52	41.23
Alaska		NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA		
Idaho		2,067	157	126	1	284	394	0	1,719	9,620	10,014	1.02	7.55		
Washington		332	2	12	0	16	257	1,644	318	2,986	4,887	6.68	159		
	X	2,399	159	138	1	298	651	1,644	2,037	12,606	14,901	1.8	14.6	0.57	1.45
National*		429,619	9,441	890	4,784	15,115	371,658	139,163	259,617	869,366	1,380,187	1.82	50.9	100	100

NA = Not available.

* Rare earths and platinum not included.

The quantities of waste generated and land-disposed in 1974 by each of the five metals mining and concentrating industries are given in Table 4 on a state, national, and EPA regional basis.

For SIC 1021 (Copper Ores), the total waste amounted to 366 million metric tons (403 million tons) of waste rock, 44.5 million metric tons (49 million tons) of overburden, and 241 million metric tons (265 million tons) of concentrator wastes (tailings).

The copper ore mining and concentrating industry accounted for 85 percent (651 million metric tons (716 million tons)) of the total national metals mining and concentrating wastes generated by the five segments examined during this study.

In SIC 1031 (Lead-Zinc Ores), the total waste quantity in 1974 was 14 million metric tons (15 million tons), consisting of 2 million metric tons (2.2 million tons) of waste rock, and 12 million metric tons (13 million tons) of tailings. The lead-zinc and zinc industries produced about 2 percent of the total national metals mining and concentrating wastes.

California was the only contributor to wastes in SIC 1092 (Mercury Ore) in 1974. There were only eight operating domestic mercury mines in 1974*, and six were located in California. The total production of waste reported was 1.6 million metric tons (1.7 million tons), consisting of 908,000 MT (1 million tons) of waste rock, 487,000 MT (536,800 tons) of overburden, and 166,000 MT (183,000 tons) of tailings. We were unable to obtain information from the mercury mines which closed down in 1974. SIC 1092 accounted for only 0.2 percent of the total national metals mining waste.

The total waste from mineral mining for SIC 1094 (Uranium-Radium-Vanadium Ores) in 1974 was 103 million metric tons (113 million tons), consisting of 2.3 million metric tons (2.5 million tons) of waste rock, 94 million metric tons (104 million tons) of overburden, and 6 million metric tons (6.6 million tons) of tailings. All the waste reported in Table 3 for Arkansas was contributed solely by the only domestic vanadium mining operation. SIC 1094 accounted for 13 percent of the total national waste from the metals mining industry.

For SIC 1099 (Antimony, Beryllium, Platinum, Rare Earths, Tin, Titanium, and Zirconium), the total waste generated in 1974 was 1,370,000 MT (1,500,000 tons), of which 1,060,000 MT (1,170,000 tons) was waste rock, and 318,000 MT (350,000 tons) was tailings. SIC 1099 accounted for only 0.19 percent of the total national waste from mineral mining and concentrating operations. We were unable to obtain information from the only platinum mine in the United States; therefore, the symbol NA is shown for all categories in Alaska.

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* Six of these mines closed in 1974, leaving only two operating mercury mines.

TABLE 4

QUANTITY OF DRY WASTES FROM METALS MINING BY STATE, EPA REGION, AND SICs 1021, 1031, 1092, 1094, AND 1099 (10^3 TPY) FOR 1974 (METRIC)

	SIC 1021 wastes				SIC 1031 wastes				SIC 1092 wastes				SIC 1094 wastes				SIC 1099 wastes			
State	Waste rock	Over-burden	Tailings	Total	Waste rock	Over-burden	Tailings	Total	Waste rock	Over-burden	Tailings	Total	Waste rock	Over-burden	Tailings	Total	Waste rock	Over-burden	Tailings	Total
Alaska	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	NA	NA	NA	NA
Arizona	235,868	30,393	109,904	376,165	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Arkansas	0	0	0	0	0	0	0	0	0	0	0	0	45	862	368	1,275	0	0	0	0
California	0	0	0	0	0	0	145	145	908	487	16	1,411	0	0	0	0	NA	NA	NA	NA
Colorado	0	0	0	0	37	0	880	917	0	0	0	0	0	0	514	514	0	0	0	0
Florida	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	33	33
Idaho	20	0	141	161	289	0	1,400	1,689	0	0	0	0	0	0	0	0	85	0	178	263
Illinois	0	0	0	0	5	0	102	107	0	0	0	0	0	0	0	0	0	0	0	0
Kentucky	0	0	0	0	0	0	0	0*	0	0	0	0	0	0	0	0	0	0	0	0
Maine	1	0	40	41	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Michigan	7,836	0	7,292	15,128	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Missouri	0	0	0	0	1,338	0	7,309	8,647	0	0	0	0	0	0	0	0	0	0	0	0
Montana	9,072	0	56,088	65,160	0	0	24	24	0	0	0	0	0	0	0	0	5	0	12	17
Nevada	36,287	1,963	12,156	50,406	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
New Jersey	0	0	0	0	0	0	125	125	0	0	0	0	0	0	0	0	0	0	0	0
New Mexico	53,870	11,793	19,233	84,896	0	0	125	125	0	0	0	0	1,127	12,172	2,449	15,748	0	0	0	0
New York	0	0	0	0	5	0	934	939	0	0	0	0	0	0	0	0	0	0	0	0
Oklahoma	8,165	181	209	8,555	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pennsylvania	0	0	0	0	0	0	406	406	0	0	0	0	0	0	0	0	0	0	0	0
Tennessee	23	0	714	737	78	0	67	155	0	0	0	0	0	0	0	0	0	0	0	0
Texas	0	0	0	0	0	0	0	0	0	0	0	0	484	7,263	367	8,114	0	0	0	0
Utah	14,515	181	35,017	49,713	66	0	118	184	0	0	0	0	38	332	420	790	961	0	95	1,056
Virginia	0	0	0	0	12	0	141	153	0	0	0	0	0	0	0	0	0	0	0	0
Washington	0	0	0	0	36	0	213	249	0	0	0	0	221	1,644	105	1,970	0	0	0	0
Wisconsin	0	0	0	0	0	0	441	441	0	0	0	0	0	0	0	0	0	0	0	0
Wyoming	0	0	0	0	0	0	0	0	0	0	0	0	353	72,224	1,860	74,437	0	0	0	0
National	365,657	44,511	240,794	650,962	1,866	0	12,430	14,296	908	487	16	1,411	2,268	94,497	6,083	102,848	1,051	0	318	1,369
Region I	1	0	40	41	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Region II	0	0	0	0	5	0	1,059	1,064	0	0	0	0	0	0	0	0	0	0	0	0
Region III	0	0	0	0	12	0	547	595	0	0	0	0	0	0	0	0	0	0	0	0
Region IV	23	0	714	737	78	0	67	155	0	0	0	0	0	0	0	0	0	0	33	33
Region V	7,836	0	7,292	15,128	5	0	543	548	0	0	0	0	0	0	0	0	0	0	0	0
Region VI	62,035	11,974	19,442	93,451	0	0	125	125	0	0	0	0	1,656	20,292	3,184	25,137	0	0	0	0
Region VII	0	0	0	0	1,338	0	7,309	8,647	0	0	0	0	0	0	0	0	0	0	0	0
Region VIII	23,587	181	91,105	114,873	103	0	1,022	1,125	0	0	0	0	391	72,556	2,794	75,741	966	0	107	1,073
Region IX	272,155	32,356	122,060	426,571	0	0	145	145	908	487	16	1,411	0	0	0	0	0	0	0	0
Region X	20	0	141	161	325	0	1,613	1,938	0	0	0	0	221	1,644	105	1,970	85	0	178	263

NA = Not available.
* Tailings sold as agricultural lime.

Potentially Hazardous Wastes

A determination was made as to how much of the waste being generated by the five metal mining industries should actually be considered potentially hazardous, taking into account available data on the following factors:

1. The constituents in the wastes.

2. Geological and mineralogical data.

3. The solubility of the metals and minerals in the waste.

4. The concentration of the metals and minerals in the waste.

5. The presence or absence of pyrite, which could cause acid leaching.

6. Background values of hazardous materials.

Table 5 presents summary data on the quantity of potentially hazardous wastes generated by all mining and concentrating in each state in 1974. In preparing these data, an identification was made as to the wastes from the mining and concentrating operations which normally contain hazardous substances in concentrations above the background levels of the land in or on which they are disposed. Such wastes were considered to be potentially hazardous. In most instances, only the concentrator tailings were found to contain hazardous materials above background levels, and therefore considered to be potentially hazardous waste. However, for uranium mining and concentrating, both mine waste rock and concentrator tailings were considered to be potentially hazardous.

For the SIC 1021 (Copper Ores) category, we concluded that the only potentially hazardous wastes are the tailings from the concentrator operations. These tailings can contain small amounts of hazardous materials at concentrations above disposal area background levels. Furthermore, these tailing wastes are vulnerable to water leaching (if pyrite is present) and wind blowing because they consist of finely divided solids. In contrast, the mining wastes (overburden and waste rock) do not contain concentrations of hazardous materials above the background levels for the soil and rock on which they are disposed, and are therefore not considered potentially hazardous.

The national total of dry potentially hazardous waste from SIC 1021 amounted to 140 million metric tons (154 million tons) in 1974. This waste consisted entirely of tailings generated in the ore concentration processes. The wet weight of potentially hazardous wastes was 461 million metric tons (508 million tons).

TABLE 5

TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING SICs 1021, 1031, 1092, 1094, AND 1099 FOR 1974 (10^3 TPY) (METRIC)

State	Region	Total process waste†	Total potentially hazardous waste†	Waste rock‡		Overburden‡		Concentrator tailings				% Potentially hazardous waste in region	
								Dry weight		Wet weight§			
				Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Dry	Wet
Maine	I	41	40	1	0	0	0	40	40	160	160	0.02	0.03
New York	II	939	934	5	0	0	0	934	934	4,670	4,670	0.58	0.86
Virginia	III	153	141	12	0	0	0	141	141	2,091	2,091	0.09	0.38
Tennessee	IV	737	714	23	0	0	0	714	714	7,483	7,483	0.45	1.38
Illinois		107	102	5	0	0	0	102	102	510	510		
Michigan		15,128	7,292	7,836	0	0	0	7,292	7,292	34,618	34,618		
Wisconsin		441	441	0	0	0	0	441	441	2,205	2,205		
	V	15,676	7,835	7,841	0	0	0	7,835	7,835	37,333	37,333	4.89	6.87
New Mexico		100,769	16,201	54,997	1,127	23,965	0	21,807	15,074	70,758	48,727		
Oklahoma		8,555	209	8.165	0	181	0	209	209	836	836		
Texas		8,114	851	484	484	7,263	0	367	367	2,276	2,276		
	VI	117,438	17,261	63,646	1,611	31,409	0	22,383	15,650	73,870	51,839	10.77	9.54
Missouri	VII	8,647	7,309	1,338	0	0	0	7,309	7,309	31,079	31,079	4.56	5.72
Colorado		1,431	1,394	37	0	0	0	1,394	1,394	9,814	9,814		
Montana		65,200	36,491	9,077	0	0	0	56,123	36,491	196,553	127,841		
Utah		51,684	23,426	15,526	38	513	0	35,645	23,388	114,337	75,212		
Wyoming		74,437	2,213	353	353	72,224	0	1,860	1,860	5,889	5,889		
	VIII	192,752	63,524	24,993	391	72,737	0	95,022	63,133	326,593	218,765	39.64	40.25
Arizona		376,165	52,167	235,868	0	30,393	0	109,904	52,167	320,274	152,021		
California*		145	145	0	0	0	0	145	145	965	965		
Nevada		50,406	7,900	36,287	0	1,963	0	12,156	7,900	37,156	24,147		
	IX	426,716	60,212	272,155	0	32,356	0	122,205	60,212	358,395	177,133	37.57	32.59
Idaho		2,144	1,750	394	0	0	0	1,750	1,750	9,973	9,973		
Washington		2,219	539	257	221	1,644	0	318	318	2,986	2,986		
	X	4,363	2,289	651	221	1,644	0	2,068	2,068	12,959	12,959	1.41	2.38
National*		767,462	160,259	370,665	2,223	138,146	0	298,611	158,036	854,633	543,512	100	100

* Rare earths and platinum not included.
† Dry weight.
‡ Dry weight = wet weight.
§ Wet weight = dry weight plus water.

In SIC 1031 (Lead-Zinc Ores), our analysis of the waste characteristics indicates that the concentration of hazardous materials in the mining wastes (waste rock and overburden) does not usually exceed the background concentrations in the native soil and rock on which they are disposed. Therefore, these mining wastes are not judged to be potentially hazardous. On the other hand, the amount of hazardous materials in the concentrator tailings is commonly in excess of background values; and, therefore, most tailings from processing lead-zinc ores are classified as potentially hazardous wastes.

The total amount of dry tailings (potentially hazardous waste) generated by SIC 1031 in 1974 was 11.8 million metric tons (13 million tons). The wet weight of potentially hazardous wastes was 61 million metric tons (67 million tons). SIC 1031 accounted for 7 percent of the total national potentially hazardous waste from metals mining in 1974.

In SIC 1092 (Mercury Ores), there are no potentially hazardous wastes generated.

The uranium-radium-vanadium ore industries (SIC 1094) produced potentially hazardous waste consisting of all uranium waste rock and tailings. The uranium wastes generally contain significant amounts of radioactive substances, such as uranium minerals, and in some cases, trace amounts of potentially hazardous materials such as thorium, radium, arsenic, and other hazardous metals.

The total potentially hazardous wastes generated by SIC 1094 in 1974 were 7,938,000 MT (8,750,000 tons), or 5 percent of the total dry national potentially hazardous wastes. This waste was comprised of 2,223,000 MT (2,450,000 tons) of waste rock from the mining uranium ore and 5,715,000 MT (6,300,000 tons) of dry uranium concentrator tailings. The wet weight of uranium concentrator tailings was 17 million metric tons (19 million tons) of potentially hazardous waste, which is 3.2 percent of the total national wet waste from this industry.

In the miscellaneous ores industry (SIC 1099), the dry weight of potentially hazardous wastes in 1974 consisted of 189,000 MT (208,000 tons) of antimony concentrator wastes in Idaho and Montana, and 90,000 MT (99,000 tons) of beryllium concentrator wastes in Utah. The wet weight of potentially hazardous wastes was 3.3 million metric tons (3.6 million tons). This total excludes the rare earth wastes in California, for which no data could be obtained. The percentage contribution of SIC 1099 to the total national dry potentially hazardous waste in 1974 was 0.19 percent, and to the total wet weight was 0.68 percent.

Tables 6 and 7 contain our projections for the total and potentially hazardous wastes to be generated and land disposed in 1977 and 1983. To arrive at the numbers in the tables, we used the projected mine production or metal demand data furnished by the U.S. Bureau of Mines.[1/] The assumption was made that the ratio of waste to ore would reamin relatively constant during this period. We further assumed that the ore grade and concentrating method would also remain as they are. Our evaluation of this industry leads us to believe that the mining and concentrating methods in use today will also be used in the immediate future.

We do not, however, believe that ore grades will remain constant, or that the ratio of waste to ore will remain constant. Ore grades are constantly changing, even within a mine. The higher grade ores will be depleted before the lower grade ores are processed. Another factor that influences the waste generated is the market price for the products of the mine and concentrator. As the price for the product falls, metal mines and concentrators are closed down. There are over 1,600 abandoned mines and concentrators in these SIC categories. Some of these were abandoned because the ore was depleted, the ore grade dropped below the economic break-even point, the market price of the product dropped below the profitable level, or transportation of raw materials and products was too expensive.

As shown in Table 6, the total potentially hazardous dry land-disposed wastes in 1977 are projected to be 187 million metric tons (206 million tons). The wet wastes are projected to be 632 million metric tons (695 million tons). These quantities represent a 16 percent increase over 1974.

As shown in Table 7, the total potentially hazardous land-disposed dry wastes in 1983 are projected to be 260 million metric tons (286 million tons). The wet wastes are projected to be 851 million metric tons (938 million tons). This represents a 58 percent increase over 1974, and 36 percent over 1977.

Waste disposal from air and water pollution control devices in the mining and concentrating of these ores did not add to the waste burden in 1974. All material collected in the air pollution control devices in the grinding of the ores was returned to the process for recovery of metal values.

In 1977 and 1983, no change in the operation of air pollution control devices is envisioned. All material collected will be returned to the process for recovery of metal values. However, there will be wastes disposed from water-pollution-control devices in 1977 and 1983. It is MRI's estimate that these quantities will not exceed 1 percent of the total wastes disposed on land.

Waste Treatment and Disposal

The waste treatment and disposal practices for potentially hazardous wastes from mining and concentrating of the metal ores of interest are described in this report.

TABLE 6

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING FOR SICs 1021, 1031, 1092, 1094, AND 1099 FOR 1977 (10^3 TPY) (METRIC)

				Waste rock‡		Overburden‡		Concentrator tailings: Dry weight		Concentrator tailings: Wet weight§		% Potentially hazardous waste in region	
State	Region	Total process waste†	Total potentially hazardous waste†	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Dry	Wet
Maine	I	46	45	1	0	0	0	45	45	181	181	0.02	0.03
New York	II	1,202	1,196	6	0	0	0	1,196	1,196	5,978	5,978	0.64	0.95
Virginia	III	195	180	15	0	0	0	180	180	2,676	2,676	0.10	0.42
Tennessee	IV	833	807	26	0	0	0	807	807	8,456	8,456	0.43	1.34
Illinois		137	131	6	0	0	0	131	131	653	653		
Michigan		17,095	8,240	8,855	0	0	0	8,240	8,240	39,118	39,118		
Wisconsin		564	564	0	0	0	0	564	564	2,822	2,822		
	V	17,796	8,935	8,861	0	0	0	8,935	8,935	42,593	42,593	4.78	6.74
New Mexico		121,966	20,160	62,725	1,852	33,325	0	25,916	18,308	83,645	58,750		
Oklahoma		9,667	236	9,226	0	205	0	236	236	945	945		
Texas		13,331	1,398	795	795	11,933	0	603	603	3,739	3,739		
	VI	144,964	21,794	72,746	2,647	45,463	0	26,755	19,147	88,329	63,434	11.66	10.04
Missouri	VII	11,068	9,356	1,713	0	0	0	9,356	9,356	39,781	39,781	5.01	6.29
Colorado		2,018	1,971	47	0	0	0	1,971	1,971	12,959	12,959		
Montana		73,648	41,207	10,256	0	0	0	63,391	41,207	221,961	144,316		
Utah		58,767	26,717	17,509	62	750	0	40,505	26,655	129,698	85,487		
Wyoming		122,300	3,636	580	580	118,664	0	3,056	3,056	9,676	9,676		
	VIII	256,733	73,531	28,392	642	119,414	0	108,923	72,889	374,294	252,438	39.34	39.94
Arizona		425,066	58,949	266,531	0	34,344	0	124,192	58,949	361,910	171,784		
California*		186	186	0	0	0	0	186	186	1,235	1,235		
Nevada		56,959	8,927	41,000	0	2,218	0	13,736	8,927	41,906	27,286		
	IX	482,211	68,062	307,531	0	36,562	0	138,114	68,062	405,131	200,305	36.42	31.69
Idaho		2,673	2,185	488	0	0	0	2,185	2,185	12,237	12,237		
Washington		3,556	809	409	363	2,701	0	446	446	3,918	3,918		
	X	6,229	2,994	897	363	2,701	0	2,631	2,631	16,155	16,155	1.60	2.56
National*		921,277	186,900	420,188	3,652	204,140	0	296,942	183,248	983,574	631,997	100	100

* Rare earths and platinum not included.
† Dry weight.
‡ Dry weight = wet weight.
§ Wet weight = dry weight plus water.

TABLE 7

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING FOR SICs 1021, 1031, 1092, 1094, AND 1099 FOR 1983 (10^3 TPY) (METRIC)

State	Region	Total process waste†	Total potentially hazardous waste†	Waste rock‡		Overburden‡		Dry weight		Wet weight§		% Potentially hazardous waste in region	
				Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Dry	Wet
Maine	I	60	59	1	0	0	0	59	59	237	237	0.02	0.03
New York	II	1,324	1,317	7	0	0	0	1,317	1,317	6,585	6,585	0.51	0.77
Virginia	III	216	199	17	0	0	0	199	199	2,948	2,948	0.08	0.35
Tennessee	IV	1,094	1,060	34	0	0	0	1,060	1,060	11,075	11,075	0.41	1.30
Illinois		151	144	7	0	0	0	144	144	719	719		
Michigan		22,390	10,792	11,598	0	0	0	10,792	10,792	51,235	51,235		
Wisconsin		622	622	0	0	0	0	622	622	3,109	3,109		
	V	23,163	11,558	11,605	0	0	0	11,558	11,558	55,063	55,063	4.45	6.47
New Mexico		195,668	34,535	84,728	4,998	71,438	0	39,502	29,537	125,136	92,531		
Oklahoma		12,660	308	12,084	0	268	0	308	308	1,237	1,237		
Texas		35,986	3,775	2,147	2,147	32,211	0	1,628	1,628	10,094	10,094		
	VI	244,314	38,618	98,959	7,145	103,917	0	41,438	31,473	136,467	103,862	14.87	12.21
Missouri	VII	12,192	10,306	1,887	0	0	0	10,306	10,306	43,821	43,821	3.97	5.15
Colorado		3,573	3,521	52	0	0	0	3,521	3,521	17,141	17,141		
Montana		96,490	54,002	13,433	0	0	0	83,057	54,002	290,853	189,159		
Utah		78,514	35,989	22,814	169	1,740	0	55,086	35,820	171,624	113,720		
Wyoming		330,128	9,815	1,566	1,566	320,313	0	8,249	8,249	26,118	26,118		
	VIII	508,705	103,327	37,865	1,735	322,053	0	149,913	101,592	505,736	346,138	39.78	40.68
Arizona		556,724	77,207	349,085	0	44,980	0	162,658	77,207	474,000	224,991		
California*		204	204	0	0	0	0	204	204	1,361	1,361		
Nevada		74,600	11,692	53,705	0	2,905	0	17,990	11,692	54,990	35,738		
	IX	631,528	89,103	402,790	0	47,885	0	180,852	89,103	530,351	262,090	34.30	30.80
Idaho		3,019	2,467	553	0	0	0	2,467	2,467	13,993	13,993		
Washington		9,088	1,746	1,031	980	7,291	0	766	766	5,006	5,006		
	X	12,107	4,213	1,584	980	7,291	0	3,233	3,233	18,999	18,999	1.62	2.23
National*		1,434,703	259,760	554,749	9,860	481,146	0	399,935	249,900	1,311,282	850,818	100	100

* Rare earths and platinum not included.

† Dry weight.

‡ Dry weight = wet weight.

§ Wet weight = dry weight plus water.

All potentially hazardous land-disposed wastes generated by this industry are disposed of by this industry on their property. No outside contractors are used and no off-site disposal is practiced.

The waste disposal practices for the potentially hazardous land-disposed wastes are described at three levels of technology. These levels are:

Level I - Technology currently employed by typical facilities, i.e., broad average present treatment and disposal practice.

Level II - Best technology currently employed. Identified technology at this level must represent the soundest process from an environmental and health standpoint, currently in use in at least one location. Installations must be commercial scale; pilot and bench scale installations are not suitable.

Level III - Technology necessary to provide adequate health and environmental protection. Level III technology may be more or less sophisticated or may be identical with Level I or II technology. At this level, identified technology may include pilot or bench scale processes providing the exact stage of development is identified.

The only potentially hazardous land-disposed waste from mining of these ores is waste rock from uranium mining. Level I technology for the disposal of waste rock from open-pit uranium mines is utilized by 90 percent of the operating mines and is briefly described in this section; a more detailed explanation is contained in Section VI. Briefly, the waste rock is co-mingled with the overburden and 93 percent (1.2 million metric tons (1.3 million tons)) is discarded on a surface waste dump. The dump surface is contoured and stabilized by vegetation. The rest of the waste rock, 7 percent (93,000 MT (102,000 tons)), is used as mine backfill.

Level II and III technologies, which are the same for disposal of waste rock from open-pit uranium mines, are the same as Level I with the following additions: the final dump is covered with soil and vegetation, and the active dump is periodically covered with overburden or soil.

Level I technology for the disposal of potentially hazardous waste rock from underground uranium mines is to pile the rock on a surface waste dump, contour, and allow natural vegetation to stabilize the dump. All underground mines are using Level I technology and about 800,000 MT (880,000 tons) are disposed of this way.

Level II and III technologies are practiced by 10 percent (12) of the uranium mines. To meet Level II and III technology requirements, the waste dump must be contoured, covered with a layer of topsoil or approved subsoil, vegetated by seeding, and fertilized to maintain a stabilizing plant cover. About 100,000 MT (110,000 tons) of waste rock are disposed in this manner.

There is one universal method for the disposal of concentrator tailings from copper, lead-zinc, beryllium, antimony, uranium, cadmium, and vanadium processes, and that is the use of a tailings pond for the land disposal of potentially hazardous wastes. Therefore, Level I technology for land disposal of all potentially hazardous concentrator wastes is the use of a tailings pond for all the metal concentrators encompassed in this study.

Level II and III technologies will vary for the different concentrator wastes. Level II technology for the land disposal of potentially hazardous wastes from copper concentrators is to compact and vegetate the dike or dam surrounding the tailings pile. At the present time, three copper companies are utilizing Level II technology for the disposal of potentially hazardous concentrator tailings. They are disposing of about 60 million metric tons (65 million tons), which is about 43 percent of the total potentially hazardous waste from copper mining and concentrating.

Level III technology for the disposal of copper concentrator wastes is to add chemicals to the effluent from the tailings pond to precipitate the metal ions, which are then returned to the tailings pond. This practice is at present followed by one operator, and a treatment plant is being built at a second concentrator. Both treatment plants will treat about 23 percent of the total effluent water from tailings ponds at copper concentrators.

Level II and III technology for the disposal of potentially hazardous wastes from concentrating lead-zinc ores is the addition of a wastewater treatment plant to precipitate the minerals from the effluent and return the solids to the pond. The process has been piloted by at least two companies.

Level II and III technologies for disposal of potentially hazardous wastes from uranium concentrating operations consists of adding the zero discharge (no effluent), and special treatment to avoid erosion losses of tailings.

Level II and III technologies for antimony and beryllium are the same as Level I. There are only one beryllium plant and two antimony plants.

There were no potentially hazardous wastes generated in the mining and concentrating of zinc, mercury, and titanium-zirconium ores, and therefore, no technology levels were identified. Tin is no longer mined in the United States, and we were able to learn nothing about the mining or concentrating of the platinum group ores or the rare earth ores.

Cost Analysis

Representative plants for all categories of mining and concentrating, in which potentially hazardous wastes were generated, were used for the purpose of the cost analysis. The capacities, handling rates, ore grade, operating hours and other items listed in the plant specifications were selected to represent average size plants in most instances. The model plant for each ore mined and concentrated is representative of the particular ore category; it is not meant to represent the entire industry. In copper, lead-zinc, and uranium-ore mining and concentrating, there are significant variations in size. The process used for copper and lead-zinc ores is flotation. There are some differences in flotation plants, depending on the other metals present in the ore. However, the basic process, equipment, and operating characteristics are similar. There are two main processes for concentrating uranium ores, and a representative plant was selected for each process.

The scope of the present study prohibited a detailed parametric study of the economic impact as a function of plant size and configuration, and the model plants used in the analysis present only a simplified flow diagram for each type of facility. As a consequence, there are some weaknesses and shortcomings in the model plant concept. Because of the limitations imposed by the model plant approach, the results of the economic analysis should be viewed as indicative of the probable economic impact, and not the absolute impact.

The cost analysis was conducted from the perspective of a mining corporation. Only those costs that the corporation pays directly (or that are "realized" by the firm) are included. External social costs, such as the value of the pollution emitted, are excluded.

The future effects of inflation are also ignored in the analysis. Keeping all estimates in terms of constant 1973 dollars enhances comparisons and avoids the necessity of predicting trends in inflation. However, any changes in the "real" (or noninflated) value of capital, labor, or resources are included.

The differential impact of taxes has also been excluded in the methodology. In other words, the costs calculated represent "before tax" estimates. Taxes enter the calculation as a separate cost item, and are expressed as a percentage of invested capital. Handling taxes in this manner helps keep the analysis easily comprehensible to a wide variety of readers. The method should also enhance comparisons with other EPA studies and private sector cost estimates. The discussion below presents details of the methodology.

Capital costs include all buildings, equipment, land, etc., that do not change with small changes in the amount of hazardous wastes disposed. These costs are often termed fixed costs.

The cost of capital equipment can be calculated in a variety of ways and with an even wider variety of assumptions. For this study, it was assumed that all capital is financed through the sale of private debt. The cost of financing with debt is assumed to be 10 percent. The 10 percent value was chosen because it was approximately the cost of private debt in 1973, and it has been used in numerous other EPA studies.[2] The use of a discount rate accounts for the interest cost of debt. Therefore, interest costs do not have to be calculated separately.

The period of analysis is equal to the useful life of the longest-lived piece of capital equipment (usually the tailings pond). Although costs are expressed in terms of dollars per metric ton of potentially hazardous wastes, the period of analysis is still important. If the period of analysis is short, relatively large capital cost items will be undervalued. This undervaluation usually occurs because the salvage value of the long-lived equipment understates the equipment's remaining usefulness. Therefore, using a short period of analysis tends to favor less capital-intensive technologies and equipment.

The nominal cost of the capital equipment, excluding land, is allocated over its useful life by the use of straight-line depreciation. It is unlikely that the true value of the asset will decrease by equal amounts each period. Nonetheless, simplicity and comparability with other EPA reports support the use of the straight-line method.

The levelized annual capital costs and the annual operation and maintenance costs, and the yearly fuel and power costs are summed. This sum represents the total yearly cost of operating a given hazardous waste disposal facility (using a prespecified technology and capacity size). The yearly quantity of hazardous waste disposed by the representative facility is obtained. The cost of disposal can then be expressed in dollars per metric ton of potentially hazardous waste.

The technique described above is used in all the cost analyses made in this study. Important assumptions are listed with the cost estimates. More than one cost estimate was made in most cases to demonstrate the range of results and the dependence of those results on input assumptions. In addition, the costs were aggregated to the entire industry level. The aggregated estimate was then compared to the costs of production for that industry.

The analyses and estimation of costs associated with the handling and disposal of wastes in the various metal mining industries relied heavily upon information provided by respondents to the AMC-mailed questionnaires and field interviews.

Cost data received from the responding companies, along with information regarding their waste quantities, characteristics, treatment processes, and handling techniques, provided the basis for developing standard unit costs for the various functional operations involved in waste disposal.

The raw cost data were related to the specific disposal operations, categorized generally as follows:

1. Wastes from the mining operation
   a. Dump for leaching or further processing
   b. Dump for disposal
   c. Leave in place
   d. Use as surface (pit) backfill
   e. Return underground as mine backfill
2. Wastes from the concentration operations
   a. Transfer to sand plant for separation
   b. Discharge to tailings pond
   c. Return underground as mine backfill
   d. Use as surface fill
   e. Filter liquids

The unit disposal processes associated with each typical mining operation (classified by type of ore, metal and mining method) were identified and disposal costs estimated on a process-by-process basis. Flow sheets, identifying the flow of all materials from the time the ore left the mine until the metal or concentrate was shipped, were employed in identifying and quantifying the appropriate disposal operations.

All cost estimates were prepared on the basis of raw material leaving the mine. Thus, waste rock and overburden left in place, either in an open pit or underground, were considered to be a cost of mining rather than a waste disposal cost.

Wide variations were reported in disposal costs by the various respondents; this is to be expected because of the widely differing conditions under which wastes are disposed. For example, surface dumping of waste rock, the most common disposal method, can vary from as little as $0.10/ton to more than $1.00/ton, depending upon the distance the waste must be hauled to the dump site. Discharging tailings as a slurry into a pond, the most common method of waste disposal from the concentration operation, is also subject to wide variations, again depending on the overall facilities layout and the amount of pumping force required. Median figures were used as much as possible, with the same unit costs applied to the same unit operations, regardless of the type of ore being handled.

The other major factor causing variations in waste disposal costs was simply the quantity of wastes associated with the different mining operations and mining methods. These variations are especially apparent when comparing disposal costs among the mining industries.

Finally, to reflect the cost impact of waste disposal in each industry, the disposal costs were related to the quantity of final output attained in each typical operation, expressing the disposal costs in dollars per ton of metal or concentrate produced or shipped.

The total costs of waste disposal associated with typical operations in each metals mining industry were estimated. The cost estimates were prepared by applying appropriate unit costs for each disposal function to the quantity of waste handled in each operation. Disposal costs were broken down between wastes from the mining operations and wastes from the concentration operations. All cost estimates were prepared on the basis of quantities of ore leaving the mine and expressed as costs per million metric tons of potentially hazardous waste.

Wide variations were found among the different industries, depending on the ore, mining methods, quantity of waste materials handled, and unit operations employed. The costs for disposal of potentially hazardous wastes from ore mining are summarized in Table 8. Level I, II, and III technologies are displayed in this table. The costs are expressed in dollars per metric ton of potentially hazardous wastes, and the total cost to the industry. The cost is also related to the value added for each type of mine. The costs for disposal of potentially hazardous wastes from concentrators are summarized in Table 9. Level I, II, and III technologies costs are displayed in this table.

The following paragraphs summarize the major conclusions of this study.

1. The five metals mining industries covered in this study generate large quantities of waste which are land-disposed. These wastes usually consist of overburden, waste rock, and tailings. The copper ore industry produces the largest amount of these wastes, followed in descending order by uranium-vanadium, lead-zinc, mercury, and miscellaneous ores.

2. In most cases, the overburden and waste rock do not contain potentially hazardous materials in concentrations greater than the land on or in which they are disposed. Therefore, these wastes generally do not add to the potential hazards of the area. The exception is uranium ore waste rock, which usually contains radioactive materials (uranium and radium) above background levels.

3. Wastes from ore concentration processes (tailings) may contain potentially hazardous substances in higher concentrations than the land on which they are disposed, and are therefore considered potentially hazardous. Furthermore, these wastes are fine-grained and subject to wind and water erosion unless properly treated.

TABLE 8

SUMMARY OF COSTS FOR LAND DISPOSAL OF POTENTIALLY HAZARDOUS WASTES FROM ORE MINING (SICs 1021, 1031, 1094, AND 1099)

		Level I technology				Levels II and III technology			
Ore mined	Type of mine	Disposal mode	Range of cost ($/MT)	Total cost to industry* ($\$10^6$/yr)	% of value added in mining[†]	Disposal mode	Range of cost ($/MT)	Total cost to industry* ($\$10^6$/yr)	% of value added in mining[†]
Uranium	Open Pit	Surface Dump	$0.004-$0.25	$.008-$.47	0%-.29%	Surface Dump	$0.20-$0.24	$.38-$.45	.23%-.28%
Uranium	Open Pit	Mine Back-fill	$0.16-$0.21	$.023-$.030	.01%-.02%	N.A.	N.A.		
Uranium	Underground	Surface Dump	$0.04-$0.25,	$.008-$.05	0%-.03%	Surface Dump	$0.19-$0.44	$.038-$.088	.02%-.05%

N.A. = Not Applicable

* Assuming all mines of that type and disposal mode utilize the stated technology level. Based on waste disposal levels in 1974.

[†] 1972 Value Added in Mining adjusted to 1973 dollars using wholesale price index. Sources: 1972 Census of Mineral Industries Bureau of Census, U.S. Govt. Printing Office, 1975 and Economic Report of the President, January 1976, U.S. Govt. Printing Office 1976.

TABLE 9

SUMMARY OF COSTS FOR LAND DISPOSAL OF POTENTIALLY HAZARDOUS WASTES FROM CONCENTRATORS (SICs 1021, 1031, 1094, and 1099)

		Level I technology				Levels II and III technology			
Ore and Concentrator	Location	Disposal mode	Range of cost ($/MT)	Total cost to industry[*]/ ($10⁶/yr)	% of value added in mining[†]	Disposal mode	Range of cost ($/MT)	Total cost to industry ($10⁶/yr)	% of value added in mining[†]
Copper Flotation	NA	Tailings Pond	$.0275 $.0282	$6.6-$6.8	.6%				
Lead-Zinc Flotation	NA	Tailings Pond	$0.55- $1.63	$6.8- $20.3	3.2%- 9.5%	Tailings Pond[#]	$1.55- $2.22	$19.2- $27.6	8.9%-12.9
Uranium Leach	Wyoming	Tailings Pond	$0.18- $0.30	$0.50-[§] $0.84	0.5%- 0.8%	Tailings Pond[#]	$0.195- $0.318	$ 0.54-[§] $ 0.89	0.57%- 0.89%
Uranium Leach	New Mexico	Tailings Pond	$0.13- $0.18	$0.36-[**] $0.51		Tailings Pond[#]	$0.134- $0.186	$ 0.38-[**] $ 0.52	
Antimony Flotation Plus	NA	Tailings Pond	$1.63- $1.65	$0.310- $0.312	< 1.1%[††]		Same as Level 1		
Beryllium Leach	NA	Tailings Poug	$0.54- $0.58	$0.05- $0.052			Same as Level 1		

	Level II technology				Level III technology			
Ore and Concentrator	Disposal mode	Range of cost ($/MT)	Total cost to industry[##] ($10⁶/yr)	% of value added in mining[†]	Disposal mode	Range of cost ($/MT)	Total cost to industry[##] ($10⁶/yr)	% of value added in mining[†]
Copper Flotation	Tailings Pond[§§]	$0.32- $0.33	$77.1- $79.4	7.2%	Tailings Pond[***]	$0.516- $0.522	$124.3- $125.7	11.3%- 11.4%

NA = Not applicable.

[*] Derived from metric tons of concentrator wastes (dry weight) per year in 1974.

[†] 1972 value added in mining adjusted to 1973 using Wholesale Price Index. Source: 1972 Census of Mineral Industries, Bureau of Census, Government Printing Office, 1975; Economic Report of the President, January 1976, U.S. Government Printing Office, 1976.

[#] Soil cover plus vegetative stabilization.

[§] Total cost to industry located in Wyoming, Utah, and Colorado in 1974.

[**] Total cost to industry located in New Mexico and Texas in 1974.

[††] Antimony and beryllium are classified in SIC 1099 but do not comprise the only metals in the category.

[##] Assuming entire copper industry utilized this technology level.

[§§] Plus vegetation stabilization.

[***] Plus vegetation stabilization and wastewater treatment plant.

4. The presence of pyrite (FeS_2) in the wastes significantly increases the hazard potential. Pyrite can react in the presence of air and water to form sulfuric acid, which can leach potentially hazardous metals from the waste. If pyrite is not present, or if the pyrite is treated with alkaline material (e.g., lime or soda ash) and the water contacting the waste is kept above pH 7.5, no leaching of potentially hazardous metals will occur. It is, therefore, important to determine whether the wastes contain free pyrite.

5. There is a general lack of available data as to the potentially hazardous substances contained in metal mining wastes. Mining companies usually analyze their land-disposed wastes primarily for the metals being extracted, not for potentially hazardous substances per se.

6. Based on the limited data presently available, potentially hazardous substances contained in land-disposed mining wastes which have been identified are as follows:

* Copper ore wastes (tailings): copper, lead, zinc, pyrite, and cadmium.

* Lead-zinc ore wastes (tailings): lead, zinc, copper, pyrite, and cadmium.

* Uranium ore wastes (waste rock): uranium and radium.

* Uranium ore wastes (tailings): radium, thorium, and arsenic.

* Antimony ore wastes (tailings): antimony, lead, and zinc.

* Beryllium ore wastes (tailings): beryllium.

7. Adequate waste treatment and land-disposal methods for potentially hazardous wastes are available to the companies mining and concentrating copper, lead-zinc, zinc, mercury, and miscellaneous metals ores.

8. There are wide variations between the various metals mining industries as to the costs of waste disposal, depending on the type of ore, mining method, quantity of waste handled, and the processes employed.

All units reported in this report are metric units with English units in parentheses in the text where possible. A table of conversion units and a glossary of terms are contained in Appendix D.

Included with this report are four appendices. Appendix A contains a list of the active mine and concentrator operations in 1974 for SICs 1021, 1031, 1092, 1094, and 1099. Appendix B contains listings of the mines and concentrators visited, mines and concentrator operators that returned the questionnaire, and a sample questionnaire. A report on the geology and mineralogy of the various mining regions is contained in Appendix C. A reference bibliography, the glossary, and a table of conversion units are contained in Appendix D.

REFERENCES

1. U.S. Department of the Interior, Bureau of Mines. Commodity data summaries, 1974; Appendix I to mining and minerals policy.

2. Gruber, G. I. Assessment of industrial hazardous practices, organic chemicals, pesticides and explosives industries. Environmental Protection Agency, Report No. 2566-60/0-TU-00, Jan. 1976.

SECTION I

COPPER ORES (SIC 1021)

Industry Characterization

History of the Industry. Copper has been a contributor to mankind's progress from the Stone Age to the Space Age. During this span of several thousand years, uses have ranged from tools, weapons, utensils, statuary, and architectural applications to the generation and transmission of electrical energy.

Copper was one of the first metals used by humans, because it was available naturally in essentially the pure form, and when beaten or forged, it hardened sufficiently to permit primitive man to fashion utensils and sharp implements and weapons of lasting quality. Little is known about prehistoric mining of copper, although it seems certain that both open-pit and underground mining methods were used.

Evidence of the first mining of copper in North America was discovered by archeologists in pits in the Upper Peninsula of Michigan and on Isle Royale in Lake Superior. Copper was first produced in the American Colonies at Simsbury, Connecticut, in 1709; and a significant ore body was found in Orange County, Vermont (Ely mine), in 1820. Discovery of the ore deposits in Michigan in the early 1840's began a new era for copper production in the United States. Extensive ore bodies were found in Montana and Arizona during the years 1860 to 1880, and smelter production from domestic ores increased from 660 MT (762 tons) in 1850 to 27,433 MT (30,175 tons) in 1880. In Bingham Canyon, Utah, the Utah Copper Company (now Utah Copper Division, Kennecott Copper Corporation) began exploitation of a large low-grade porphyry deposit in 1906. The success of this operation stimulated search for and development of other large deposits of similar characteristics both in the United States and abroad. Development of the froth flotation process in the first two decades of the 20th century vastly increased the availability of copper contained in the large low-grade deposits which constitute most of the U.S. copper reserves.

The United States has been the largest copper-producing nation in the world since 1883, and in 1973 produced about 22 percent of the total world copper. Other principal copper-producing nations are the U.S.S.R., Canada, Zambia, Chile, Peru, and Zaire.

Domestic Production and Capacity. During the period 1963 to 1973, U.S. copper output (recoverable content of ore*), increased 41 percent from 1,100,000 MT (1,212,542 tons) annually to 1,550,000 MT (1,708,582 tons). During the same period, world production of copper increased 55 percent from 4,489,000 MT (4,948,275 tons) to 6,984,000 MT (7,698,542 tons) (Table 10). While the U.S. government statistics are not available for 1974, it is estimated that the amount of copper concentrate produced in the United States was nearly 7,324,456 MT† (8,073,830 tons) (Table 11). Table 12 gives copper ore production by year, and Table 13 gives the estimated copper ore production by state and region for 1974.

Domestic copper demand is forecast to increase at an annual average growth rate of 4 percent through 1983. New mine projects and expansion plans are in approximate balance with the forecast consumption increase and should maintain the present high degree of self-sufficiency. Based on the projected rate of demand, copper production in the United States would be approximately 1,822,000 MT (2,008,411 tons) of copper in 1977 and 2,306,000 MT (2,541,929 tons) in 1983.

Copper production is concentrated in comparatively few mines. In 1972, 25 mines accounted for 93 percent of the U.S. output; the five largest provided 41 percent of domestic mine production. In 1974, approximately 47 mining operations, encompassing 56 mines, were producing copper in the United States.

Mine production of copper in the United States was approximately 88 percent of the estimated capacity in 1972. Mine production capacity is expected to have increased by 16 percent between 1972 and 1975, to a total of 1,995,000 MT (2,199,000 tons) (Table 14).

Number, Location, Age, and Size of Active Mines and Mills. Ninety-eight percent of the copper production in the United States in 1974 was located in six states; Arizona, Utah, New Mexico, Montana, Nevada, and Michigan, and virtually all of the remainder in Tennessee, Maine, Idaho, Missouri, and Oklahoma (Figure 1). Forty-eight of the 57 copper mines operating in 1974 were located in the first six states (Table 15 and Appendix A).

Copper mines vary in size from small operations employing fewer than 20 workers to mines employing more than 2,500 workers. Employment data from 57 active mines in 1974 indicate that more than one-half of these operations employed more than 500 workers. More than 20 percent of the mine operations employed more than 1,000 workers.

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* Recoverable content of ore = the amount of pure copper recovered.

† Recovered copper concentrate = the concentrated copper ore before smelting and refining. The copper concentrate product will vary from 25 to 90 percent copper depending on the method used in the concentration process. It is estimated that 95 percent of the product is 25 percent copper concentrate.

TABLE 10

COPPER PRODUCTION, 1963-1983*
(1,000 MT)

Year	United States	Rest of world	Total
1963	1,100	3,389	4,489
1964	1,131	3,506	4,637
1965	1,226	3,598	4,824
1966	1,296	3,679	4,975
1967	865	3,873	4,738
1968	1,093	4,024	5,117
1969	1,401	4,244	5,645
1970	1,560	4,461	6,021
1971	1,380	4,654	6,034
1972	1,510	5,124	6,634
1973	1,550	5,427	6,984
1977	1,822†	--	--
1983	2,306†	--	--

Source: References 1 and 2.

* Data published in tons and calculated to metric tons.

† Estimated production.

TABLE 11

U.S. COPPER CONCENTRATE PRODUCTION, 1974*
(RECOVERED COPPER CONCENTRATE PRODUCT)

State	MT
Arizona	4,876,625
Idaho	17,353
Maine	3,654
Michigan	223,167
Montana	290,299
Nevada	357,498
New Mexico	616,480
Oklahoma	4,445
Tennessee	72,575
Utah	862,359
U.S. total	7,324,456

EPA regions	MT
I	3,654
II	--
III	--
IV	72,575
V	223,167
VI	620,925
VII	--
VIII	1,152,658
IX	5,234,123
X	--

Source: Reference 3.

* *Engineering & Mining Journal* and unpublished mining company data.

† Data published in tons and calculated to metric tons.

TABLE 12

TOTAL COPPER MINE PRODUCTION OF ORE BY YEAR*
(ORE PRODUCED)

Year	1,000 MT
1967	115,272
1968	154,270
1969	202,984
1970	233,807
1971	220,133
1972	242,065
1973	263,081

Source: References 3 and 4.

* Data published in tons and calculated to metric tons.

TABLE 13

U.S. COPPER ORE PRODUCTION FROM MINES BY STATE AND EPA REGION, 1974 (ORE MINED)

State*	1,000 MT
Arizona	148,417
Idaho	364
Maine	48
Michigan	8,059
Montana	56,454
Nevada	12,294
New Mexico	18,829
Oklahoma	213
Tennessee	1,996
Utah	135,552
U.S. total	382,226†
EPA regions	**1,000 MT**
I	48
II	--
III	--
IV	1,996
V	8,059
VI	19,042
VII	--
VIII	192,006
IX	160,711
X	364
U.S. total	382,226

Source: Reference 5. Engineering & Mining Journal and unpublished mining company data.

* States listed are the only ones that have copper mines.

† Incomplete total--cannot estimate quantity of ore in in situ leach.

TABLE 14

PROJECTED U.S. COPPER CAPACITY, 1972-1975
(1,000 MT)*

	1972	1973	1974	1975
Mine and concentrator	1,723	1,814	1,905	1,995
Smelter	1,778	1,814	1,877	1,995
Refinery	2,639	2,639	2,676	2,839

Source: Reference 1.

* Data published in tons and calculated to metric tons.

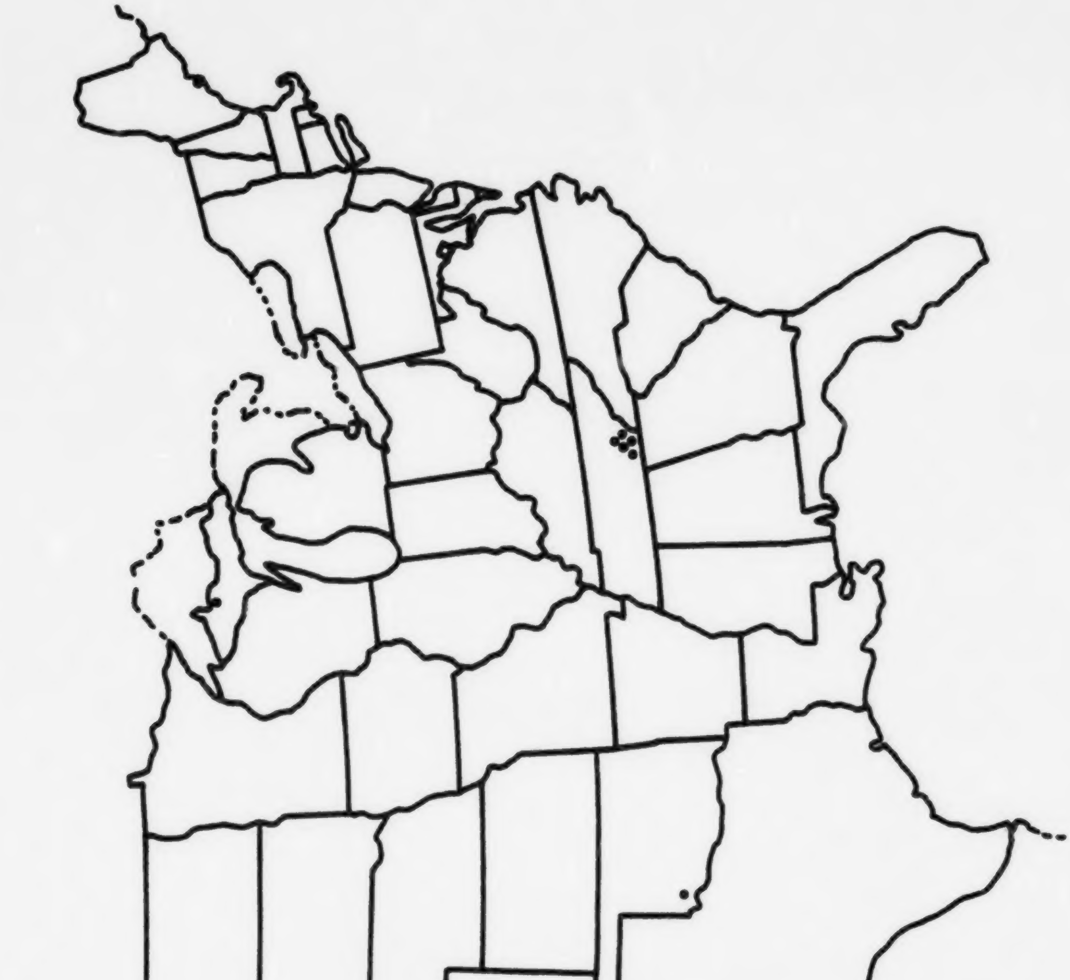


Figure 1. Location of copper (SIC 1021) mines for 1974.

TABLE 15

NUMBER, LOCATION, AND SIZE OF ACTIVE COPPER MINING AND CONCENTRATING OPERATIONS* (1974)

State	Employees								Mining operations*	No. of mines	No. of concentrators
	1-19	20-49	50-99	100-249	250-499	500-999	1,000-2,499	2,500+			
Arizona	2	2		4	2	11	6		27	31	35
Idaho				1					1	1	1
Maine			1						1	1	1
Michigan							1		1	1	1
Montana								1	1	2	2
Nevada				1		1	1		3	3	6
New Mexico		3		2	1	2			8	9	8
Oklahoma			1						1	1	1
Tennessee						1			1	5	1
Utah	1	1						1	3	3	5
U.S. total	3	6	2	8	3	15	8	2	47	57	61
EPA regions											
Region I			1						1	1	1
Region II											
Region III											
Region IV						1			1	5	1
Region V							1		1	1	1
Region VI		3	1	2	1	2			9	10	9
Region VII											
Region VIII	1	1						2	4	5	7
Region IX	2	2		5	2	12	7		30	34	41
Region X				1					1	1	1

Source: References 4 and 5.

* Location of mining company operations--may or may not include more than one mine at the location designated.

Determination of the age of active mines in the copper industry is difficult. Secondary source information does not usually give these data, and only fragmentary statistics are available. Some mines have opened and closed on many occasions. Some mines started from a multiplicity of small mines and have grown into one large mine. Where data exist, it is apparent that many active mines have been operating for a long period of time. For example, the Copper Queen operation at Bisbee, Arizona, opened in 1885, the Copperhill operation in Tennessee in 1899, and the Utah Copper operation in 1906. Likewise, there has been some recent copper mine development, such as the Pinto Valley operation and Oracle operations in Arizona which opened in 1974, and the San Xavier, Arizona, operations in 1973. Of the 30 active copper mining operations for which data are available, six were opened before 1930 and seven were opened since 1969 (Appendix A).

Employment. Total employment in active copper mines in 1974 is estimated at 34,158. About 53 percent of the copper mine workers in the United States are employed in Arizona where more than one-half of all copper mines and most of the large ore-producing mines are located (Table 16).

By-Product/Coproduct Relationships. About 98 percent of the U.S. production of copper is recovered from ore mines primarily for copper content, with the remainder being recovered from complex or base metal ores, as in the recovery of copper from the lead-zinc ores of Missouri. In addition to copper, important quantities of gold, silver, molybdenum, nickel, selenium, tellurium, arsenic, rhenium, iron, lead, zinc, sulfur, and platinum-group metals are recovered as by-products from copper ores.

Lead, molybdenum, iron, and zinc minerals are separated from copper minerals by selective flotation. Gold, silver, nickel, platinum, palladium, selenium, and tellurium are recovered from anode sludges at electrolytic copper refineries. Arsenic and sulfur are extracted during copper smelting, and rhenium is obtained in the processing of molybdenum concentrates recovered as a coproduct in the treatment of some copper ores. Copper by-products and coproduct relationships are identified in Table 17.

TABLE 16

EMPLOYMENT AT ACTIVE COPPER MINE OPERATIONS
(1974)

State	Employment
Arizona	18,090
Idaho	200
Maine	91
Michigan	2,444
Montana	2,644
Nevada	3,199
New Mexico	1,820
Oklahoma	75
Tennessee	903
Utah	4,692
U.S. total	34,158
EPA regions	**Employment**
I	91
II	--
III	--
IV	903
V	2,444
VI	1,895
VII	--
VIII	7,336
IX	21,289
X	200

Source: References 5 and 6.

TABLE 17

DOMESTIC COPPER BY-PRODUCT AND COPRODUCT RELATIONSHIPS (1972) (METRIC)*

Source	By- or coproduct	Quantity (MT)	% of Total product output
Copper	Arsenic	W	100.0
Copper	Rhenium	2.772	100.0
Copper	Selenium	0.322	100.0
Copper	Palladium	W	100.0
Copper	Tellurium	W	
Copper	Platinum	W	W
Copper	Silver	0.440	39.4
Copper	Molybdenum	18,375	36.0
Copper	Gold	0.0145	33.4
Copper	Nickel	2,272	15.9
Copper	Sulfur	533,000	5.8
Copper	Zinc	6,300	1.5
Copper	Iron	W	W
Copper	Lead	900	0.2
Copper	Copper	1,485,000	98.4
Lead	Copper	11,700	0.8
Zinc	Copper	7,200	0.5
Iron	Copper	W	W
Silver	Copper	1,800	0.1
Gold	Copper	--	W
Tungsten	Copper	--	W
Fluorine	Copper	†	W

Source: Reference 1.

* Data given in English units and calculated to metric units.

† Less than 1/2 MT.

W = Withheld to avoid disclosure.

Waste Generation and Characterization

Waste Generation. Tables 18 and 19 are compilations of the copper production and waste generation data for the United States in 1974. A total of 383,133,000 MT (422,331,835 tons) of ore and 7,410,379 MT (8,168,545 tons) of copper concentrate was produced. There was a total of 626,921,000 MT (691,062,000 tons) of dry land disposed solid waste generated from the mining and concentrating of copper ores, 365,787,000 MT (403,211,000 tons) of waste rock, 44,511,000 MT (49,066,000 tons) of overburden, 216,623,000 MT (238,785,000 tons) of concentrator tailings, and 531,578,300 MT (585,955,000 tons) of water to tailings ponds.

The ratios of total waste rock, overburden, and concentrator wastes to ore mined by state and region are shown in Table 20. In 1974, 0.95 ton of waste rock was generated and land-disposed for every ton of ore mined. Also, 0.12 ton of overburden was disposed of for every ton of ore. Nationwide, 0.63 ton of concentrator dry wastes and 2.0 tons of wet waste were generated for each ton of copper ore mined. A regional variation from 0.01 ton (EPA Region IV) to 3.26 tons (EPA Region VI) of waste rock for each ton of ore was noted. Some EPA Regions (I, IV, V, and X) had no overburden because all mines in these Regions are underground mines. The ratio of overburden to ore varied from 0.0009 ton/ton in EPA Region VIII to 0.63 ton/ton in EPA Region VI. There was not as great an EPA Regional variation in the ratio of concentrator dry and wet wastes to ore.

Three factors are involved in the amount of waste per ton of ore: type of mine, underground or open pit; ore grade; and ore type; oxide, sulfide, or mixed oxide-sulfide. An underground mine will have less waste rock and no overburden to remove and dispose on land. In general, the ore grade is higher in underground mines, hence less concentrator waste. The ore type will determine the concentrating method used. Concentration by flotation generates more solid waste than does concentration by leaching and precipitation.

Table 21 shows the potentially hazardous wastes disposed on land for 1974. Table 22 contains the projected data for 1977 and Table 23 the projected total potentially hazardous waste for 1983. The three tables show the potentially hazardous material destined for land disposal by state, EPA Region, and nationwide for the copper concentrators. Table 21 indicates that in 1974 there were 461 million metric tons (507 million tons) of potentially hazardous wet waste disposed of on land and 140 million metric tons (154 million tons) of potentially hazardous dry waste. The total process waste was 1,423 million metric tons (1,566 million tons), of this total, 772 million metric tons (849 million tons) or 54 percent of the total waste was concentrator waste and disposed of on land. The potentially hazardous material was 65 percent of the total material disposed in tailings ponds.

In 1977 there will be 521 million metric tons (573 million tons) of potentially hazardous materials for land disposal, representing an increase of 13 percent over 1974.

TABLE 18

TOTAL PRODUCTION STATISTICS BY STATE AND EPA REGION FOR SIC 1021 COPPER ORES FOR 1974 (METRIC)

					Concentrator wastes									
State	Region	Ore mined 10^3 TPY	Waste rock 10^3 TPY	Overburden 10^3 TPY	Dry weight 10^3 TPY	Wet weight 10^3 TPY	Copper conc. 10^3 TPY	Metal conc. 10^3 TPY	Miscellaneous TPY	Total TPY	Ratio of dry weight waste to ore	Ratio of dry weight waste to product	% Total dry waste in region	% Total wet waste in region
Maine	I	48	1	0	40	160	3,654	3,322	0	6,976	0.85	5.9	0.01	0.02
Tennessee	IV	1,996	23	0	714†	7,483	72,575	834,610	0	907,185	0.37	0.8	0.11	0.97
Michigan	V	8,059	7,836	0	7,292†	34,618	246,163	0	0	246,163	1.88	61.0	2.32	4.48
New Mexico		19,899	53,870	11,793	19,233†	62,933	616,480	0	0	616,480	4.3	138	-	-
Oklahoma		213	8,165	181	209	836	4,445	0	0	4,445	40.2	1,924	-	-
Total	VI	20,112	62,035	11,974	19,442	63,769	620,925	0	0	620,925	4.64	150	14.36	8.26
Montana		56,454	9,072	0	56,088	196,308	365,596	91	98	365,785	1.15	178	-	-
Utah		135,552	14,515	181	35,017	111,776	862,359	10,655	226,796	1,099,810	0.37	45	-	-
Total	VIII	192,006	23,587	181	91,105	308,084	1,227,955	10,746	226,894	1,465,595	0.6	78	17.65	39.89
Nevada		12,294	36,287	1,963	12,156	37,156	357,498	2	0	357,500	4.1	140	-	-
Arizona		148,417*	235,868	30,393	109,904†	320,274	4,876,625	55,645	0	4,932,270	2.6	80	-	-
Total	IX	160,711	272,155	32,356	122,060	357,430	5,234,123	55,647	0	5,289,770	2.7	80	65.53	46.28
Idaho	X	201	20	0	141†	828	5,024	0	0	5,024	0.8	32	0.02	0.11
National		383,133*	365,657	44,511	240,794	772,372	7,410,379	904,325	226,894	8,641,638	1.7	75		

Source: References 2, 5, and 15.

* Incomplete total cannot estimate ore for in situ mining.

† Part of tailings are classified and loose sand returned to underground mine for backfilling.

TABLE 19

CONCENTRATOR WASTES, ORE MINED AND COPPER PRODUCED FOR SIC 1021
(METRIC)

Process code*	Key**	Number of mines	Ore mined 10^3 TPY	Percent total ore	Copper† produced TPY	Percent total copper	Concentrator waste 10^3 TPY	Percent of total waste	Ratio of waste to copper produced	Ratio of waste to ore mined	Potentially hazardous waste 10^3 TPY	Percent of total waste potentially hazardous
A	A	23	149,313	38.97	1,070,224	39.04	140,179	64.71	131	0.94	140,179	64.71
B	B	12	65,807	17.18	526,458	19.21	61,494	28.39	117	0.93	-	
R	C + H	7	104,155	27.19	824,459	30.08	-	-	-	-	-	
S	D + H	1	9,181	2.40	64,265	2.34	9,101	4.20	142	0.99	-	
L	D + I	3	5,895	1.54	47,159	1.72	5,866	2.71	124	0.99	-	
M	E + K	3	7,519	1.96	37,594	1.37	-	-	-	-	-	
N	E + H	11	16,134	4.21	80,671	2.94	-	-	-	-	-	
O	G + H	2	25,129	6.56	75,387	2.75	-	-	-	-	-	
P	F + H	5	-	-	10,786	0.39	-	-	-	-	-	
Q	F + K	1	-	-	4,082	0.15	-	-	-	-	-	
	Total	68‡	383,133§		2,741,085		216,640		79	0.57	140,179	64.71

Source: References 2, 5, and 15.

* Letter denotes process described in Table 24.

† Copper produced is expressed as tons of pure (100%) copper.

‡ Duplication; some mines use more than one process.

§ Incomplete total; cannot estimate ore quantity in in situ leach.

** See Table 24.

TABLE 20

RATIO OF TOTAL WASTE ROCK-OVERBURDEN-CONCENTRATOR DRY AND WET WASTES TO ORE MINED FOR SIC 1021 COPPER ORES (METRIC)*

State	Region	Ore mined 10^3 TPY	Waste rock 10^3 TPY	Ratio of total waste rock to ore	Over-burden 10^3 TPY	Ratio of total over-burden to ore	Dry concen-trator waste 10^3 TPY	Ratio of dry concen-trator waste to ore	Wet concentrator waste 10^3 TPY	Ratio of wet waste to ore
Maine	I	48	1	0.02	0	0	40	0.8	160	3.3
Tennessee	IV	1,996	23	0.01	0	0	714	0.36	7,483	3.7
Michigan	V	8,059	7,836	0.97	0	0	7,292	0.91	34,618	4.3
New Mexico		19,899	53,870	2.7	11,793	0.6	19,233	0.97	62,933	3.2
Oklahoma		213	8,165	38.3	181	0.85	209	0.98	836	3.9
Total	VI	20,112	62,035	3.26	11,974	0.63	19,442	0.97	63,769	3.2
Montana		56,454	9,072	0.16	0	0	56,088	0.99	196,308	3.5
Utah		135,552	14,515	0.11	181	0.001	35,017	0.26	111,776	0.8
Total	VIII	192,006	23,587	0.12	181	0.0009	91,105	0.47	308,084	1.6
Nevada		12,294	36,287	2.95	1,963	0.16	12,156	0.99	37,156	3.0
Arizona		148,417	235,868	1.59	30,393	0.20	109,904	0.74	320,274	2.2
Total	IX	160,711	272,155	1.69	32,356	0.20	122,060	0.76	357,430	2.2
Idaho	X	201	20	0.1	0	0	141	0.70	828	4.1
National		383,133	365,657	0.95	44,511	0.12	240,794	0.63	772,372	2.0

* Table compiled from data in Table 18, p. 38.

TABLE 21

TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING COPPER ORES FOR SIC 1021 FOR 1974 (METRIC)*

		Dry waste weight (10^3 TPY)								Wet weight (10^3 TPY)	
				Waste rock		Overburden		Concentrator tailings		Concentrator tailings	
State	Region	Total process waste	Total potentially hazardous waste	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous
Maine	I	41	40	1	0	0	0	40	40	160	160
Tennessee	IV	737	737	23	0	0	0	714	714	7,483	7,483
Michigan	V	15,128	7,292	7,836	0	0	0	7,292	7,292	34,618	34,618
New Mexico		84,896	12,500	53,870	0	11,793	0	19,233	12,500	62,933	40,902
Oklahoma		8,555	209	8,165	0	181	0	209	209	836	836
Total	VI	93,451	12,709	62,035	0	11,974	0	19,442	12,709	63,769	41,738
Montana		65,160	36,456	9,072	0	0	0	56,088	36,456	196,308	127,596
Utah		49,713	22,760	14,515	0	181	0	35,017	22,760	111,776	72,651
Total	VIII	114,873	59,216	23,587	0	181	0	91,105	59,216	308,084	200,247
Nevada		50,406	7,900	36,287	0	1,963	0	12,156	7,900	37,156	24,147
Arizona		376,165	52,167	235,868	0	30,393	0	109,904	52,167	320,274	152,021
Total	IX	426,571	60,067	272,155	0	32,356	0	122,060	60,067	357,430	176,168
Idaho	X	161	141	20	0	0	0	141	141	828	828
National		650,962	140,179	365,657	0	44,511	0	240,794	140,179	772,372	461,242

Source: Reference 15.
* MRI engineering judgment.

TABLE 22

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING OF COPPER ORES FOR SIC 1021 FOR 1977 (METRIC)*

		Dry waste weights (10^3 TPY)								Wet weight (10^3 TPY)	
				Waste rock		Overburden		Concentrator tailings		Concentrator tailings	
State	Region	Total process waste	Total potentially hazardous waste	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous
Maine	I	46	45	1	0	0	0	45	45	181	181
Tennessee	IV	833	807	26	0	0	0	807	807	8,456	8,456
Michigan	V	17,095	8,240	8,855	0	0	0	8,240	8,240	39,118	39,118
New Mexico		95,932	14,125	60,873	0	13,326	0	21,733	14,125	71,114	46,219
Oklahoma		9,667	236	9,226	0	205	0	236	236	945	945
Total	VI	105,599	14,361	70,099	0	13,531	0	21,969	14,361	72,059	47,164
Montana		73,631	41,195	10,251	0	0	0	63,379	41,195	221,828	144,183
Utah		56,176	25,719	16,402	0	205	0	39,569	25,719	126,307	82,096
Total	VIII	129,807	66,914	26,653	0	205	0	102,948	66,914	348,135	226,279
Nevada		56,959	8,927	41,000	0	2,218	0	13,736	8,927	41,986	27,286
Arizona		425,066	58,949	266,531	0	34,344	0	124,192	58,949	361,910	171,784
Total	IX	482,025	67,876	307,531	0	36,562	0	137,928	67,876	403,896	199,070
Idaho	X	182	159	23	0	0	0	159	159	936	936
National		735,587	158,402	413,188	0	50,298	0	272,096	158,402	872,781	521,204

Source: Reference 2.

* MRI engineering judgment.

TABLE 23

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING COPPER ORES FOR SIC 1021 FOR 1983 (METRIC)*

				Waste rock		Overburden		Concentrator tailings weight (10^3 TPY)			
								Dry weight		Wet weight	
State	Region	Total process wastes	Total potentially hazardous waste	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous
Maine	I	60	59	1	0	0	0	59	59	237	237
Tennessee	IV	1,094	1,060	34	0	0	0	1,060	1,060	11,075	11,075
Michigan	V	22,390	10,792	11,598	0	0	0	10,792	10,792	51,235	51,235
New Mexico		125,650	18,500	79,730	0	17,455	0	28,465	18,500	93,140	60,535
Oklahoma		12,660	308	12,084	0	268	0	308	308	1,237	1,237
Total	VI	138,310	18,808	91,814	0	17,723	0	28,773	18,808	94,377	61,772
Montana		96,437	53,955	13,427	0	0	0	83,010	53,955	290,536	188,842
Utah		73,575	33,685	21,482	0	268	0	51,825	33,685	165,428	107,524
Total	VIII	170,012	87,640	34,909	0	268	0	134,835	87,640	455,964	296,366
Nevada		74,600	11,692	53,705	0	2,905	0	17,990	11,692	54,990	35,738
Arizona		556,724	77,207	349,085	0	44,980	0	162,658	77,207	474,000	224,991
Total	IX	631,324	88,899	402,790	0	47,885	0	180,648	88,899	528,990	260,729
Idaho	X	238	209	30	0	0	0	209	209	1,225	1,225
National		963,428	207,467	541,176	0	65,876	0	356,376	207,467	1,143,103	682,639

Source: Reference 2.

* MRI engineering judgment.

In 1983, there will be 683 million metric tons (751 million tons) of potentially hazardous materials for land disposal. This amount of potentially hazardous material represents an increase of 30 percent over 1977 and 48 percent over 1974.

Mining Processes. The copper mining industry uses a variety of mining methods. There are three general types of copper mines: open pit, underground, and in situ.[7]

Open-Pit Mining. There are several different mining methods used in open-pit mining of copper ores. The most common method is the bench method, in which drilling and blasting are carried out on one level or bench, and the drilling equipment moved to a lower bench for drilling and blasting. After the ore is loosened, it is loaded by shovels into trucks or railcars for delivery to the concentrator. At least one mine uses rippers to remove the ore from the ground and loads it into scrapers for delivery to the concentrator. Some of the smaller open-pit mines drill and blast the ore loose from the surroundings. They do not use benches but take everything to the same level.

Typically, open-pit mining involves four basic steps: drilling, blasting, loading, and hauling. The conventional mining operations include drilling, blasting, power shovel loading, and truck and rail haulage. The flow diagram for a typical open-pit mining operation is shown in Figure 2. About 55 percent of the mines include crushing and grinding at the mine. Ten percent do not crush or grind. The other 35 percent ship directly to the concentrator where crushing and grinding are carried out.

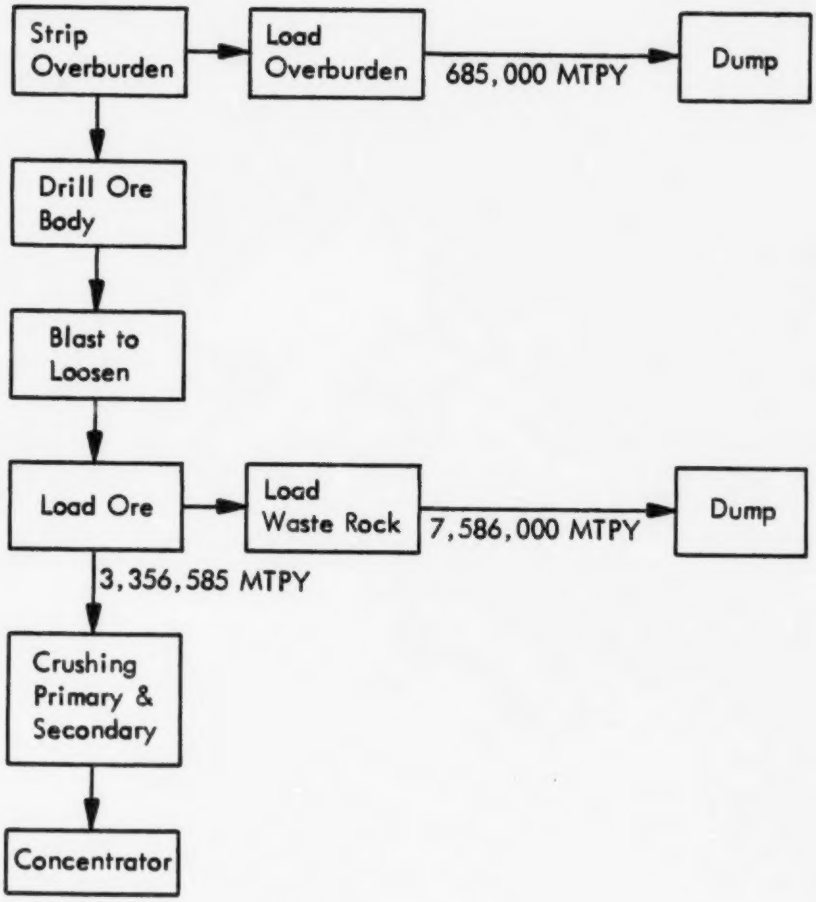
Trucks are utilized for transport at the mine site (e.g., to a primary crushing plant and concentrator), and rail haulage is used to move the ore to off-site concentrator plants. There are 40 open-pit copper mines, and they account for 85 percent of the copper ore mined.

Mass balance data for the typical mine are given in Figure 2.

Description of Individual Waste Streams. All waste rock contains less than 0.05 percent copper and is hauled to a waste rock dump. The overburden, 685,000 MT/year (754,870 tons/year), is loaded on trucks and hauled to a waste dump.

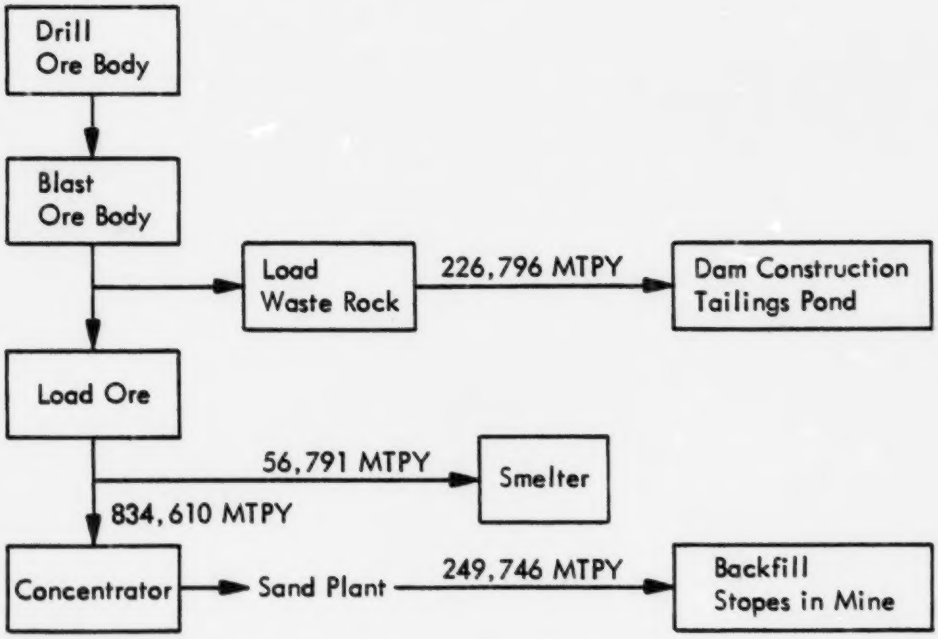
Identification of Potentially Hazardous Waste. None of the waste rock disposed on land is considered to be potentially hazardous.

Underground Mining. An underground mining method used in an Arizona copper mine is described herein as a typical example of such mining practice in the United States (Figure 3). There are 12 underground copper mines accounting for 14 percent of the copper ore mined.


Strip Overburden
Load Overburden
685,000 MTPY
Dump
Drill Ore Body
Blast to Loosen
Load Ore
Load Waste Rock
7,586,000 MTPY
Dump
3,356,585 MTPY
Crushing Primary & Secondary
Concentrator

Source: Reference 15.

Figure 2. Typical open-pit copper mine.


Drill
Ore Body
Blast
Ore Body
Load
Waste Rock
226,796 MTPY
Dam Construction
Tailings Pond
Load Ore
56,791 MTPY
Smelter
834,610 MTPY
Concentrator
Sand Plant
249,746 MTPY
Backfill
Stopes in Mine

Source: Reference 15.

Figure 3. Typical underground copper mine.

The ore at this mine occurs in a mesothermal deposit, and although the bulk of the ore has come from a vein, some of the recent production has come from replacement deposits in limestone.[8/] The ore bodies are "typical" of mesothermal copper deposits.

The principal minerals in the vein are chalcopyrite, bornite, enargite, tennantite, chalcocite, and digenite. The chief gangue minerals are pyrite, quartz, limestone, and hematite.

Description of an Underground Mining Operation. Mining is conducted using stope and pillar methods with drilling and blasting. There are several mining levels and exhaust ventilation shafts. The lowest operating level is about 1,130 m under the surface. The broken ore is raised by a skip hoist and transported by rail to a nearby concentrator. Waste rock from the mine has been used for dam construction (tailings pond). Figure 4, a flow diagram of an underground copper mining operation, indicates that part of the ore from the mine is shipped directly to a smelter.

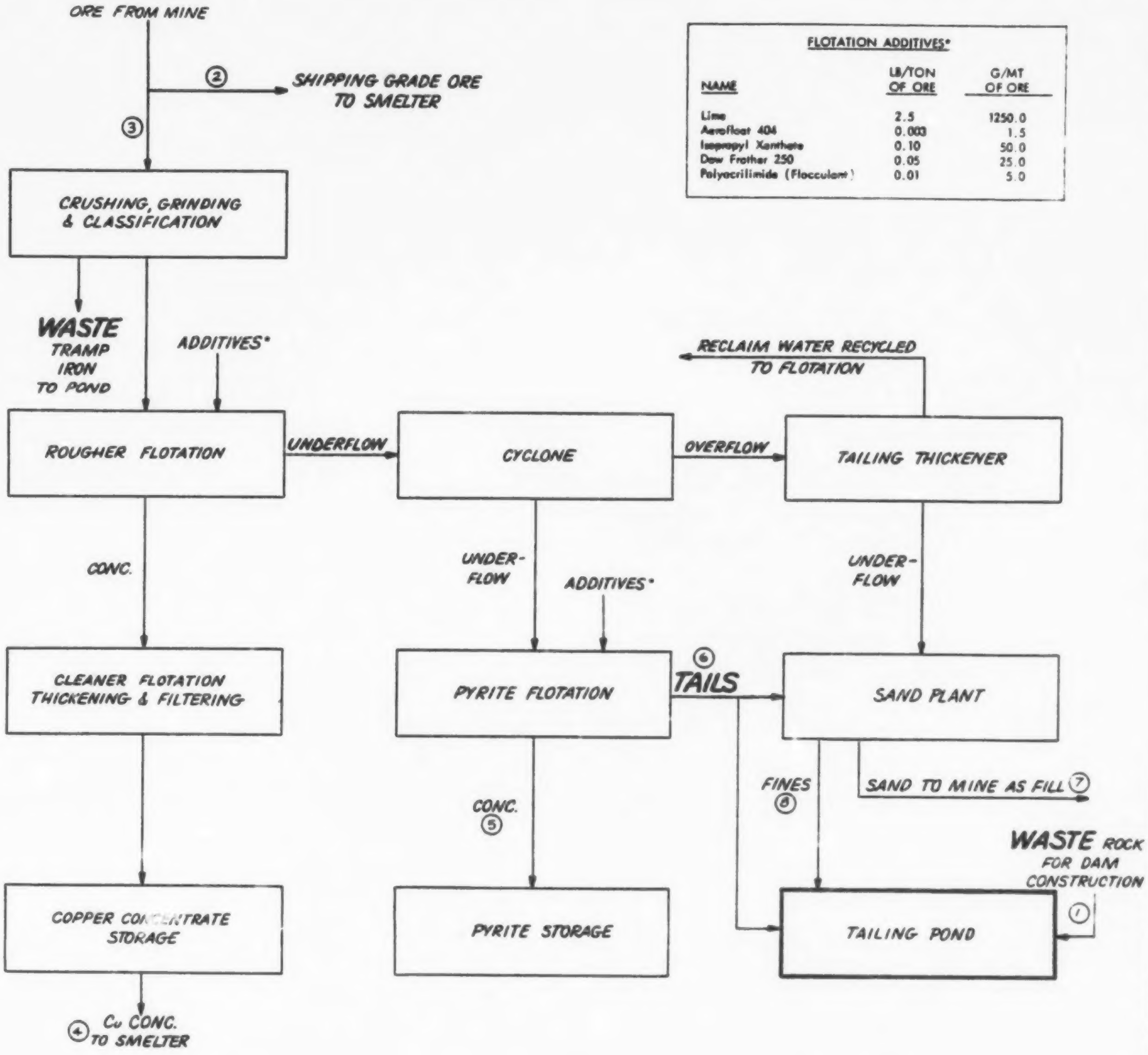
Description of Waste Streams. The only waste material in the mining operation is waste rock.

The waste rock consists principally of granite, schist, and limestone containing quartz and hematite, and consists mainly of large lumps of material along with some coarse rock particles.

Identification of Potentially Hazardous Waste Streams. Waste rock, which is mainly granite and schist from underground mining operations, is not considered to be a potentially hazardous material, because it is not subject to wind transport or water leaching of any environmental pollutants.

In Situ Mining. There are three methods used for in situ mining. One method is to remove overburden and stockpile it; the ore body is then drilled and blasted in place. A second method is to drill and blast the ore body on the side of a hill and use the blast to move the ore into an adjacent valley. The third method is to drill and blast underground to shatter the ore body. Then the ore is leached by sulfuric acid which is pumped to a concentrating plant for recovery of copper. One mine has used the first two methods, and the other two mines use the third method.

Concentrator Processes. Copper concentrating is accomplished using a primary process or a primary process in combination with secondary processes. Table 24 shows the combinations that are used in the concentrators for copper recovery. The choice of concentrating method is dictated by the ore type, size of ore body, and concentration of copper in the ore. It is completely independent of the mining method used to recover ore from the earth.


ORE FROM MINE
② SHIPPING GRADE ORE TO SMELTER
③
CRUSHING, GRINDING & CLASSIFICATION
WASTE TRAMP IRON TO POND
ADDITIVES*
ROUGHER FLOTATION
UNDERFLOW
CYCLONE
OVERFLOW
TAILING THICKENER
RECLAIM WATER RECYCLED TO FLOTATION
CONC.
UNDER-FLOW
ADDITIVES*
UNDER-FLOW
CLEANER FLOTATION THICKENING & FILTERING
PYRITE FLOTATION
⑥ TAILS
SAND PLANT
CONC. ⑤
FINES ⑧
SAND TO MINE AS FILL ⑦
WASTE ROCK FOR DAM CONSTRUCTION
①
COPPER CONCENTRATE STORAGE
PYRITE STORAGE
TAILING POND
④ Cu CONC. TO SMELTER

FLOTATION ADDITIVES*

NAME	LB/TON OF ORE	G/MT OF ORE
Lime	2.5	1250.0
Aerofloat 404	0.003	1.5
Isopropyl Xanthate	0.10	50.0
Dow Frother 250	0.05	25.0
Polyacrilimide (Flocculant)	0.01	5.0

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS	① WASTE ROCK TO DAM CONST.	② SMELTER GRADE ORE	③ ORE TO MILL	④ Cu CONC.	⑤ PYRITE CONC.	⑥ DRY TAILS TO POND	⑦ DRY TAILS TO SAND FILL	⑧ WATER TO POND
Quantity, TPY	250.000	62.800	920.000	105.000	1.700	488.000	325,300	4 930.000
Metric TPY	226.796	56.971	834.610	95.254	1.542	442.706	295,109	[illegible] 470.000

Source: Reference 15.

Figure 4. Concentrating of copper.

TABLE 24

CONCENTRATING PROCESSES FOR COPPER ORES

Ore type	Process code	Process	Waste material	Potentially hazardous waste
Sulfide-pyrite	A	Flotation	Yes	Yes
Sulfide-pyrite	B	Flotation	Yes	Yes
Sulfide-pyrite	C	Dump leach	No	No
Oxide	D	Vat leach	Yes	No
Oxide	E	Heap leach	No	No
Oxide	F	In situ leach	No	No
Oxide	G	Dump leach	No	No
Both	H	Iron precipitation	Yes	No
Both	I	Electrowinning	No	No
Both	J	Solvent extraction	No	No
Oxide	K	I + J	No	No
Oxide	L	D + I	No	No
Oxide	M	E + K	No	No
Oxide	N	E + H	Yes	No
Oxide	O	G + H	Yes	No
Oxide	P	F + H	Yes	No
Oxide	Q	F + K	No	No
Sulfide-pyrite	R	C + H	No	No
Oxide	S	D + H	Yes	No

Source: References 5 and 15.

There are two primary concentrator processes for recovery of copper from ore: flotation and leaching. Leaching is accomplished in four different modes: heap leach, dump leach, vat leach, and in situ leach. In addition to the above leaching processes, there are three secondary copper recovery processes: iron precipitation, solvent extraction, and electrowinning. The copper-rich leach solution must be further processed at the mine. Two methods are used to precipitate the copper from the leach solution. Precipitation over iron, or cementation, is the most widely used process, and the product (80 percent copper) is shipped to a smelter. The use of electrowinning to recover a 99 percent copper product is growing, however, and this material is then sold to a refinery or copper product manufacturer. The flotation product is shipped to a smelter. Smelting is the secondary recovery process for flotation concentrate. (Since smelting is not a mining or concentrating process, it is not discussed here.)

Primary Processes.

Flotation. The wastes from the copper flotation process are considered tailings, which is a fine sand-like material. Practically all copper flotation plants dispose of their tailings in a tailings pond.

An analysis of the tailings waste from a copper concentrator operation in the Coeur d'Alene Region in Idaho is shown in Table 25.[9/] These data show that the concentration of cadmium, copper, lead, and zinc (all of which are potentially hazardous metals) in the tailings stream discharged to a tailings pond are well above the background concentrations of these metals. The background data were compiled by analyzing soil samples taken in the area around the mine, concentrator, and tailings pond.

TABLE 25

ANALYSIS OF TAILINGS FROM A COPPER CONCENTRATOR

Element	Concentration, ppm	
	Concentrator	Background
Calcium	1,172	1,500
Cadmium	1.4	-
Copper	2,179	21
Iron	264,667	11,800
Potassium	115	1,800
Magnesium	6,051	3,700
Manganese	19,129	490
Sodium	75	151
Lead	1,349	51
Antimony	462	-
Zinc	868	150

Source: Reference 9.

The potentially hazardous waste from the flotation of copper sulfide ores is the tailings material. The potentially hazardous wastes in the tailings shown in Table 26 include cadmium, copper, lead, and zinc.

The hazardous substances contained in tailings from copper ore flotation operations can cause environmental pollution problems if the waste is not retained at on-site waste disposal impoundments.

TABLE 26

ANALYTICAL DATA FOR TAILINGS SOLIDS FOR SIC 1021, COPPER

	Concentration (ppm unless indicated)														
Copper	750	700	2,600	930	1,900	2,000	500	1,000	2,100	1,500	1,300	1,850	1,300	1,800	2,500
Molybdenum	0.47														
Sulfur	8,000														
Iron	29,000														
Gold (oz/ton)	0.005														
Silver (oz/ton)	0.02														
Rhenium (oz/ton)	0.004														
Aluminum	7,000														
Arsenic	-	Nil									5				
Cadmium	5,000												60		
Lead	-	200									7		100		
Magnesium	12,000														
Phosphorus	10,000														
Potassium	50,000														
Silicon	32,000														
Sodium	20,000														
Titanium	10,000														
Zinc	-	2,300									30		30	1,500	2,000
Zirconium	5,000														
Cyanides		Nil													
Mercury											0.01				
Selenium											< 1				
Chromium													290		

Source: References 9 and 15.
* Data are from 15 mines and represent 32 percent of the mines.

Description of a Flotation Process. The products of copper ore flotation concentrators are a copper concentrate and by-product concentrates. Crude ore is crushed in primary and secondary crushers to prepare a feed material for ball mill grinding. A magnetic head pulley is used on a crushed ore belt conveyor to separate pieces of scrap iron from the ore. Fine ore from the crushing operation is wet ground in a ball mill operation in closed circuit with a cyclone classifier to produce a grind of 88 percent -100 mesh ore for use in flotation.

The overflow from the cyclone classifier is sent to a bank of rougher flotation cells. Froth from these rougher cells, containing most of the copper present in the ore, is processed through cleaner type flotation cells, thickening, and filtration steps to yield a copper concentrate (25 percent copper) which is shipped to a smelter.

Underflow from the rougher cells is sent to a cyclone classifier, and cyclone overflow containing fine sands is sent to tailings thickeners which separate some reclaim water from the fines for reuse in the process (thickener underflow). The cyclone underflow is sent to by-product flotation cells. Concentrate from these cells is by-product metal concentrate. Depending on the ore produced, the by-product concentrates could be molybdenum, pyrite, or zinc. Tailings from these flotation cells are sent to a sand plant. The tailings are classified into two different size fractions in the sand plant. The coarse fraction is used to build tailings dams or to backfill underground mines. The fine fraction is discharged to a tailings pond.

A description of the types and quantities of flotation additives used in the process is given in the flow diagram in Figure 4.

The annual quantities of ore consumed, products, and wastes for this example process are shown in Figure 4. The overall recovery of copper from ore is about 95 percent. The consumption of 834,610 MT/year (920,000 tons/year) of ore corresponds to the production of 95,254 MT/year (104,999 tons/year) of copper concentrate, 1,542 MT/year (1,699 tons/year) of pyrite concentrate, and a total of 738,000 MT/year (813,000 tons/year) of tailings.

Description of Individual Waste Streams. Small quantities of scrap iron waste are recovered in the flotation operation. The scrap metal could be broken tools or pieces of equipment. This metal is sold as scrap iron or disposed of as a waste material on a waste dump.

Tailings are the only significant waste material formed in the concentrating operations. The tailings in the example described above contain about 0.2 percent copper and 0.15 percent zinc, plus nearly all the gangue material (e.g., quartz and hematite) in the feed ore used in the process. If the crude ore contains arsenic minerals, it is probable that the tailings also contain traces of arsenic; however, we could find no data to confirm the presence of arsenic in tailings.

Leaching. The commercially used leaching methods are heap leaching, vat leaching, in situ leaching, and dump leaching. There are 33 concentrators leaching copper ore, and they produce about 42 percent (1.1 million metric tons (1.21 million tons)) of the copper.

Heap Leaching. An open-pit ore deposit in Arizona is used as an example of this type of operation. This ore body consists of precipitated oxide copper minerals in host rocks of schist and granite.[10] The principal minerals are copper oxides--chrysocolla, azurite, and malachite. Only trace amounts of sulfide minerals have been identified in this ore deposit.

Ore is hauled to the leach heap site and dumped in a long thin layer along an advancing crest. The heap ore is usually started from a beginning roadway along one side of a canyon. After the ore is spread, a road grader windrows the ore over the edge to create a gradually advancing crest as more ore is hauled to the heap. By continually dumping the ore along this crest and blading it over and down the slope, a 6.1-m (20-ft) layer or lift of ore is carried across the canyon or across the top of a previously leached lift. Each heap leach is built up in 6.1-m (20-ft) lifts, and a complete heap has 10 lifts for a total height of 61 m (200 ft). However, leaching is started before a heap is completed.

When a lift is completed, the top 0.3 m (1 ft) is densely compacted from the travel of the equipment across the surface. To remove this compacted layer, a ripper breaks it up. Then a scraper is brought in, and the top 0.3 to 0.45 m (1-1.5 ft) is picked up and dumped over the nearest crest. This operation also helps to level the heap to a common elevation. Then the heap is ripped about 0.9 m (3 ft) deep to provide a porous top layer.

Under the original heaps, the canyon bottoms and side walls were sealed with a 10-cm (4-in.) layer of soil cement to prevent loss of copper-rich solution to the groundwater. This method of providing water-tight basins was generally satisfactory, but as the weight of millions of tons of ore accumulated in the heaps, it was noticed that some cracking and pulling of the soil barriers were occurring, and their integrity became doubtful.

Laboratory tests of the "topsoil" of decomposed granite from this site showed that if the soil was compacted at 10 to 11 percent moisture to 98 to 100 percent Proctor density, it became as impermeable as an adobe brick when contacted with dilute acid liquors. For all the new heap areas built from 1968 to date, the canyon bottoms and walls have been compacted very thoroughly, with water added to achieve the proper moisture content, to prepare them for solution collection, and to prevent loss of copper-bearing acid solution to the groundwater.

In the canyon bottoms and part way up the walls, collecting berms and coffer dams are built to collect the solutions flowing from the leach heaps. Perforated pipes covered with washed gravel are used to form a drain to collect the pregnant solutions in each collection area. The pregnant solution is drained to a pond for storage and to control feed to the precipitation plant.

At present there are 20 to 22 leach dumps at this site, but not all dumps are complete. A complete heap, 61 m (200 ft), is leached for 4 months, rested for 4 months, and leached again. Mass balance data are shown in Figure 5.

Until a heap is abandoned, there is no waste from the leaching operation. After the leaching of the ore is terminated, the spent rock residue represents a potentially hazardous waste material which could cause environmental pollution through continued natural leaching by rainfall and runoff unless the residual acid is washed out by water flooding of the heap. Some states require the filing of an action plan that contains the provision that heaps must be water flooded before abandonment.

At the present time, four concentrators are using heap leaching.

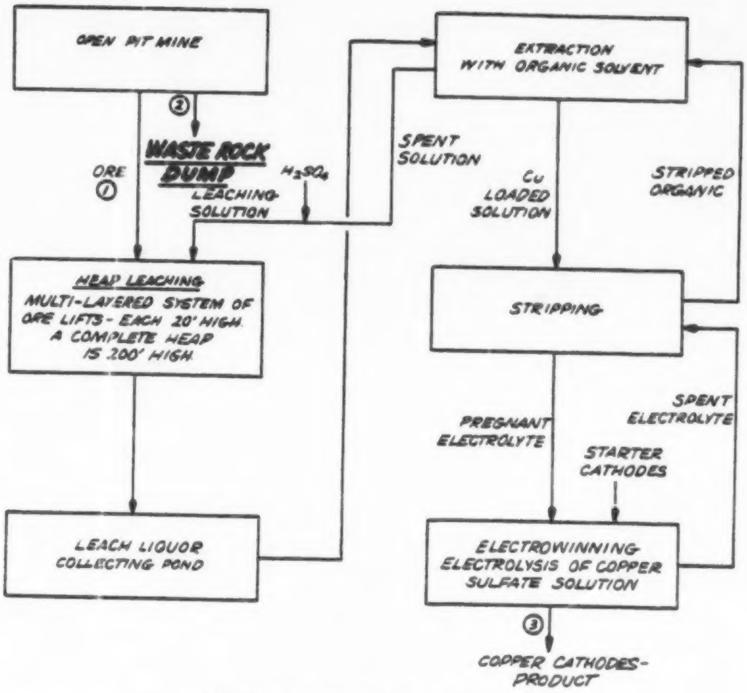
Dump Leaching.[11/] A typical dump leaching and copper precipitation operation is described herein as an example of this method for recovering cement copper from the ore. The flow diagram is shown in Figure 6.

The copper ore recovered from open-pit mining and used for dump leaching in the example case is a highly altered quartz-monzonite porphyry. The principal copper mineral is chalcocite, which is deposited in many areas of the mine as a thin coating on pyrite crystals. The copper content of the ore ranges up to about 1 percent.

Ore from the open-pit mine is transported by dump trucks to a dump. The ore may or may not be crushed. The criteria for crushing are the ore size and contact area available for leaching with the sulfuric acid solution. About 34 percent (129,284,000 MT/year (142,220,000 tons/year)) of the copper ore is processed by this method, and 33 percent (900,000 MT/year (990,000 tons/year)) of the copper metal is recovered.

Leaching at this example dump was started in 1956, and has proceeded simultaneously with dump construction over the years up to 1974. Former leach areas and haulage surfaces have been ripped with D-9 tractors before deposition of succeeding layers of ore. The dump surface is prepared for leaching by ripping, followed by construction of a grid of 3- by 3-m (10- by 10-ft) ponds on the surface of the leach dump and suitable access roadways.

Make-up water from underground wells combined with barren solution from copper precipitation is pumped to the dumps through an epoxy-lined pipeline and distributed through 10- to 20-cm diameter polypropylene lines. Distribution of the water to the surface of the dump is controlled by use of pinch valves on the polypropylene pipe.


OPEN PIT MINE
②
WASTE ROCK DUMP
ORE
①
LEACHING SOLUTION
H_2SO_4
SPENT SOLUTION
EXTRACTION WITH ORGANIC SOLVENT
Cu LOADED SOLUTION
STRIPPED ORGANIC
HEAP LEACHING
MULTI-LAYERED SYSTEM OF ORE LIFTS - EACH 20' HIGH. A COMPLETE HEAP IS 200' HIGH
STRIPPING
PREGNANT ELECTROLYTE
SPENT ELECTROLYTE
STARTER CATHODES
LEACH LIQUOR COLLECTING POND
ELECTROWINNING ELECTROLYSIS OF COPPER SULFATE SOLUTION
③
COPPER CATHODES-PRODUCT

MATERIALS BALANCE AND COMPOSITIONS

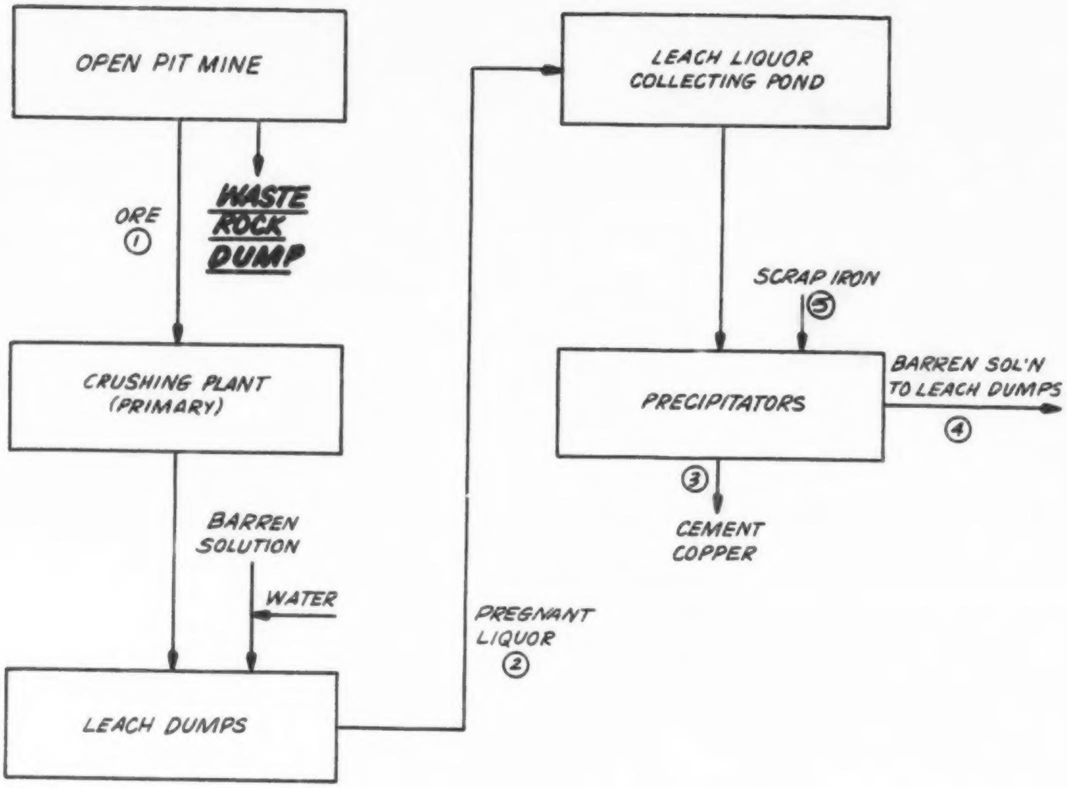
DATA ITEMS	① CRUDE ORE	② WASTE ROCK	③ COPPER CATHODE PRODUCT
Quantity, TPY	4,200,000	5,400,000	36,925
Metric TPY	3,810,177	4,898,794	33,498
Assay, %Cu	0.47	0.07	99.9+

TYPICAL METALLURGICAL DATA

	FLOW				ASSAYS G/L	
	LITERS/MIN.	GPM	Cu.	ACID	Fe	Fe'''
Leach Solution	4,939	1305	0.75	6.9	1.0	0.9
S-X Plant Feed	3,671	970	2.90	3.8	1.0	0.9
Raffinate	—	—	0.30	7.8	—	—
Loaded Organic	10,503	2775	1.01	—	—	—
Stripped Organic	—	—	.09	—	—	—
Pregnant Electrolyte	2,612	690	36.20	145.0	1.5	1.3
Spent Electrolyte	—	—	32.50	150.7	—	—
Current Efficiency				84.25		
Pounds of Acid Used				150,650		
Pounds of Acid per Pound Net Elect. Copper				5.10		

Source: Reference 15.

Figure 5. Concentrating of copper ore, heap leaching and electrowinning.


OPEN PIT MINE
WASTE ROCK DUMP
ORE ①
CRUSHING PLANT (PRIMARY)
BARREN SOLUTION
WATER
LEACH DUMPS
PREGNANT LIQUOR ②
LEACH LIQUOR COLLECTING POND
SCRAP IRON ⑤
PRECIPITATORS
BARREN SOL'N TO LEACH DUMPS ④
③ CEMENT COPPER

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS	① CRUDE ORE	② PREGNANT LIQUOR	③ CEMENT COPPER	④ BARREN SOL'N	⑤ SCRAP IRON
Quantity, TPY	15,000,000		13,911		27,820
Metric TPY	13,607,800		12,620		25,240
Assay, % Cu	0.1 - 0.6		80*		
gpl Cu		3 - 5		0.1 - 0.5	
pH		2.5 - 3.5		4 - 4.2	

Source: Reference 15.

*80% Cu on a drug basis. Product normally contains 15 - 20% moisture.

Figure 6. Concentrating of copper ore by dump leaching and precipitation.

Water is added to the surface of the leach dumps for 2 to 3 months and then discontinued for a rest period of approximately 1 year. This operation is then repeated. The period of time used in applying leach water is determined by productivity, with longer periods being used in areas that tend to produce higher grade leach liquor. During the water addition period, the water is pumped continuously to the leach dump at a rate of about 7,570 liters/min (2,000 gpm). The pyrite contained in the leach dump reacts with percolating leach water and oxygen from air to generate sulfuric acid and ferric sulfate. The resulting solution, thus formed, then dissolves copper and other constituents from the dump material. The dump is self-sustaining in regard to sulfuric acid production, and no acid is added. If oxide ore is processed, sulfuric acid is added to the solution before it is distributed on the leach dump. The copper oxide ore in leach dumps amounts to 25,129,000 MT/year (27,700,000 tons/year) or about 20 percent of the total copper ore processed by leaching.

The effluent liquor (pregnant liquor) which drains from the base of the leach dump is collected in ponds constructed at points of natural drainage egress. This pregnant liquor from the ponds is pumped to a copper precipitation plant.

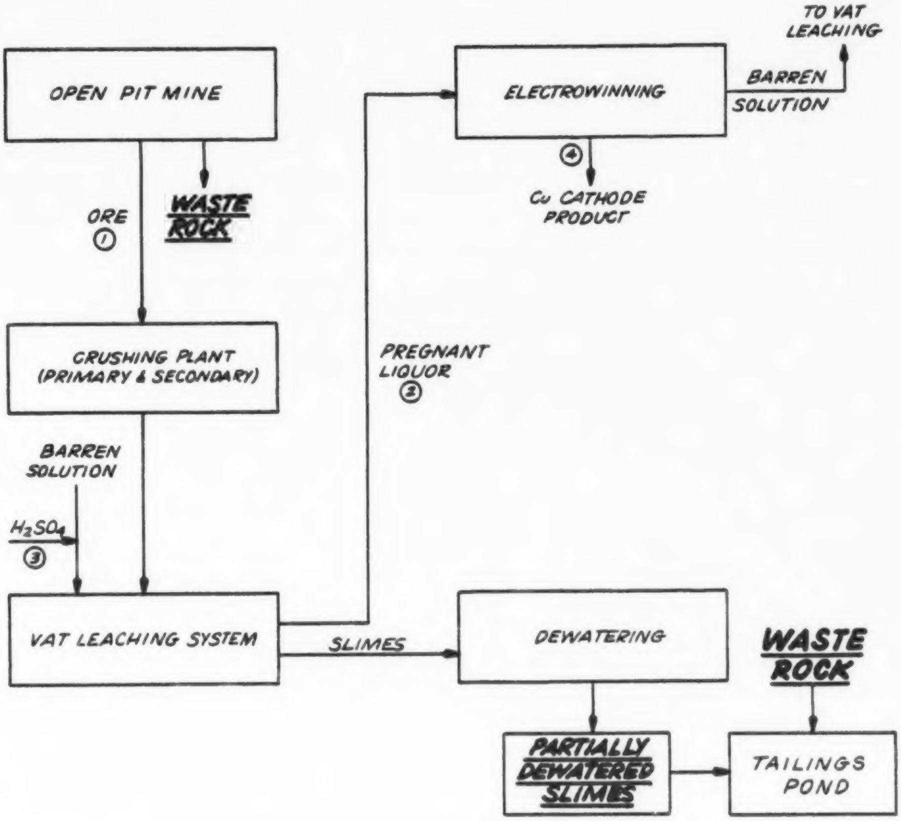
Some problems have been encountered in this example leach dump, all of which are probably typical of most dump leaching operations. These problems are: (1) a gradually declining copper content in the effluent liquor, despite continuous addition of new ore to the leach dump; (2) difficulty in obtaining water penetration of some areas of the dump as indicated by a tendency for the water to migrate laterally in the upper layers of the dump; (3) the unknown location and effect of deposits of iron salts in the dump.

The available production data given in Figure 6 show the mass balance of materials.

The dump leach pile is not considered to be waste material until the leaching operation is terminated. When leaching is stopped, proper treatment and disposal practices must be carried out to prevent environmental pollution from the inactive dump. The State of Arizona requires the filing of a plan of action for the company to follow before closing the leach dump.

Vat Leaching. A flow sheet for a typical vat leaching and electrowinning operation is given in Figure 7.

Crushed oxide ore is leached in a series of open-top concrete vats with an aqueous solution of sulfuric acid (20-25 g/liter H_2SO_4). The leach solution is used at ambient temperature, and the pH is maintained at 1 to 1.5. The ore is leached for 14 days in each vat. Then the slimes are separated and dewatered by a thickening operation and pumped to a canyon disposal dump which is also used for disposal of waste rock containing limestone, less than 0.04 percent copper, and no pyrite. The waste rock is not considered to be potentially hazardous. The copper-rich leach solution is pumped to an electrolytic copper recovery plant.


OPEN PIT MINE
WASTE ROCK
ORE
①
ELECTROWINNING
BARREN SOLUTION
TO VAT LEACHING
④
Cu CATHODE PRODUCT
CRUSHING PLANT (PRIMARY & SECONDARY)
PREGNANT LIQUOR
②
BARREN SOLUTION
H_2SO_4
③
VAT LEACHING SYSTEM
SLIMES
DEWATERING
WASTE ROCK
PARTIALLY DEWATERED SLIMES
TAILINGS POND

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS	① CRUDE ORE	② PREGNANT LIQUOR	③ LEACH LIQUOR	④ COPPER CATHODE PRODUCT	⑤ SLIMES DRY WEIGHT	⑥ WATER WEIGHT
Quantity, TPY	3,700,000			25,550	3,775,400	12,880,000
Metric TPY	3,356,585			23,179	3,333,400	14,200,000
Assay, % Cu	0.9			99.5+		
gpl Cu		30	5			
gpl H_2SO_4			20			

Source: Reference 15.

Figure 7. Concentrating of copper ore by vat leaching and electrowinning.

Electrolytic Recovery of Copper. Pregnant liquor, containing 30 to 35 g/liter of copper, is sent to electrolysis cells. The electrolysis cell operation reduces the copper concentration in the liquor from about 30 to 35 g/liter down to 5 g/liter as the copper is plated onto copper starter cathodes. The operating liquor temperature in the electrowinning steps is held at about 27°C by steam heat. The copper cathode product contains 99.5+ percent copper.[12]

Barren solution from the electrowinning operation is combined with sulfuric acid and used as leach solution in the vat leaching operation.

The available production data for this example process presented in Figure 7 show the mass balance of materials. The only wastes are the partially dewatered slimes which are produced in the vat leaching step and sent to an on-site tailings pond. These slimes are not considered to be a potentially hazardous waste since they contain only oxidized copper and may only be leached by acid. There are no other waste streams in the concentrating operations.

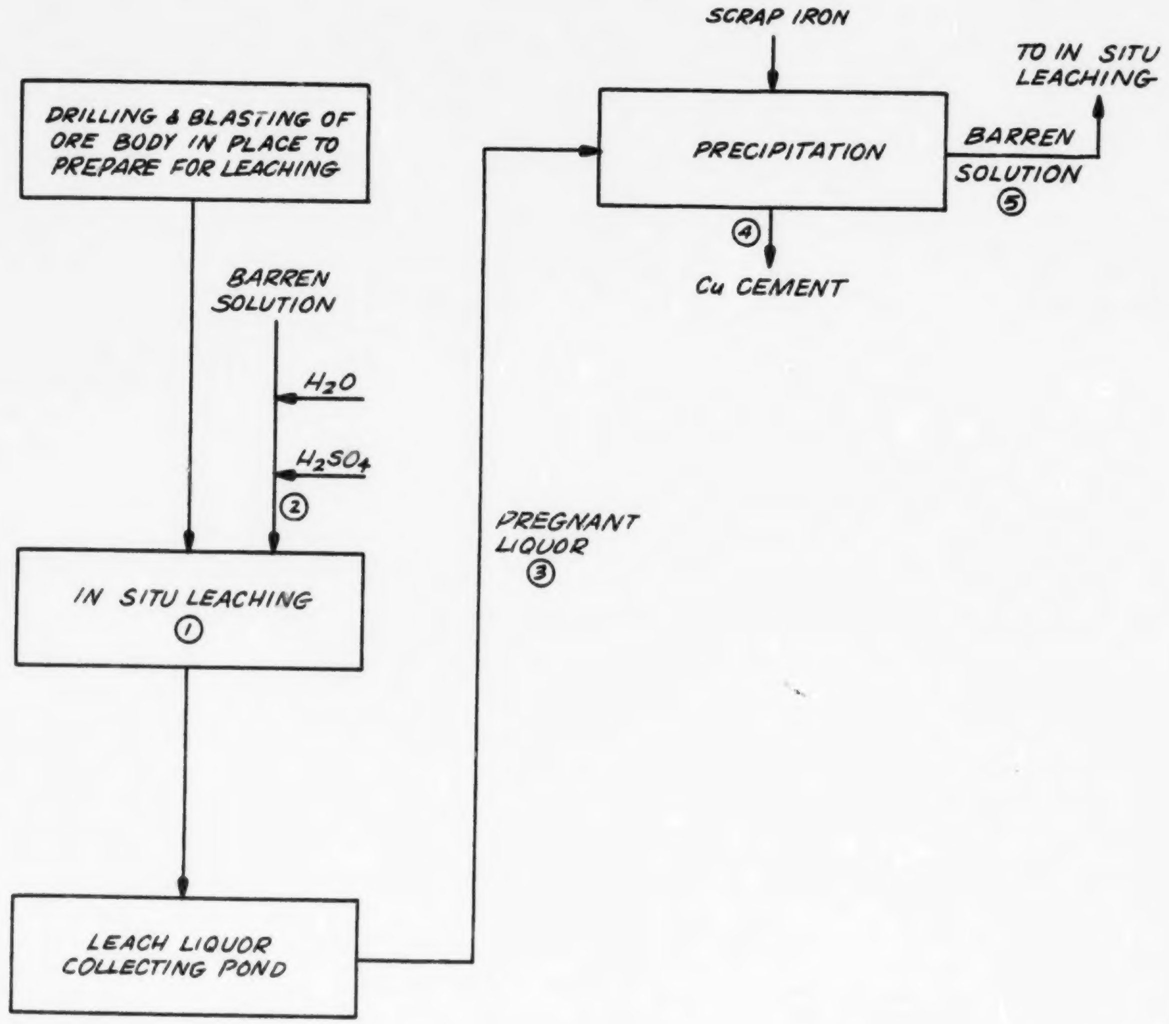
In Situ Leaching.[13] In 1974, there were three domestic copper mines which were using in situ leaching operations. There were 14,868 MT (16,390 tons) of copper concentrate produced, about 0.5 percent of the total copper produced. In addition, three other mining companies are known to be doing experimental work on in situ leaching of copper ore.

One in situ leaching and precipitation method currently being used at an Arizona copper mine is described herein as an example of this type of operation. The ore is a silicate copper, chrysocolla ($CuSiO_3 \cdot 2H_2O$) and malachite ($CuCO_3 \cdot Cu(OH)_2$). No sulfide minerals are present in this ore body. The host rock is granite and schist, which if left undistrubed, is impervious to water and acid. The copper content of this ore ranges from 0.2 to 0.6 percent. The flow diagram for this process is shown in Figure 8.

Elimination of conventional mining processing by in situ leaching results in lower capital and operating costs. Steps eliminated include loading, transporting, crushing, grinding, waste and tailings disposal, and land reclamation.

In order to provide permeability of the copper ore deposit, the first step in the process involves drilling and in-place blasting of the deposit. This operation fractures and shatters the ore and prepares the ore body for the percolating solution which must contact the ore.

In the example mine, the host rock (granite and schist) forms barriers at the perimeter of the ore zone impervious to water and acid, i.e., a solution control barrier.


SCRAP IRON
TO IN SITU LEACHING
DRILLING & BLASTING OF ORE BODY IN PLACE TO PREPARE FOR LEACHING
PRECIPITATION
BARREN SOLUTION ⑤
④
Cu CEMENT
BARREN SOLUTION
H_2O
H_2SO_4
②
PREGNANT LIQUOR ③
IN SITU LEACHING ①
LEACH LIQUOR COLLECTING POND

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS	① CRUDE ORE	② LEACH LIQUOR	③ PREGNANT LIQUOR	④ Cu CEMENT	⑤ BARREN SOL'N
Assay, gpl Cu		0.2 - 0.4	3 - 6		0.2 - 0.5
% Cu	0.2 - 0.6			80	
pH		1.2 - 1.7	2.5 - 2.8		4.5

Source: Reference 15.

Figure 8. Concentrating of copper ore by in situ leaching and precipitation.

Following the in-place blasting of the ore, a leaching solution containing 2.5 to 5 percent H_2SO_4 (pH of 1.2 to 1.7) is sprayed onto the surface of the broken ore and allowed to percolate downward through the ore. Process water is obtained from wells in the area. The pregnant copper solution from the in-place leaching operation is collected in a pond at the base of the leaching area. This pregnant liquor is pumped as required to a precipitation plant.

At the precipitation plant, the pregnant solution containing copper sulfate is contacted in cone precipitators with scrap iron and chemically detinned iron to precipitate copper cement from the solution. The chemical reaction for this precipitation is:

$$CuSO_4 + Fe \longrightarrow FeSO_4 + Cu$$

The cement copper is filtered on a vacuum filter and then further dewatered by drainage on covered concrete slabs. The dewatered product (80 percent copper) is shipped to a smelter.

Barren solution from the precipitation step is recycled to the leaching site. Prior to being sprayed onto the surface of the ore bed, the barren liquor is mixed with sulfuric acid and makeup water to provide the desired composition in the leach solution. The available data for this example process are insufficient to permit a mass balance. However, the available operating process data are shown in Figure 8.

There are no wastes at an operating in situ site and precipitation plant. When all the leaching operations are finally terminated, the in situ leached ore becomes a waste. In situ leaching of a given site lasts for many years. A thorough water rinsing of the leach area to remove the acid and recover the copper in the acid solution is planned by the operating companies.

Secondary Processes. There are four secondary processes for recovery of copper from leach solutions. All of these methods can be used with all of the primary processes. The selection of the secondary process is guided by the results of research and development by the individual mining companies and the Bureau of Mines.

Precipitation of Cement Copper.[14/] In the copper concentrating industry, the three types of precipitation equipment used are cone precipitators (conical-shaped vertical vessels), launders consisting of a series of open-top concrete tanks, and vat precipitators.

In the example case, the pregnant liquor is precipitated on a continuous basis in concrete launders. The pregnant liquor is contacted with scrap iron in the launders to accomplish the following precipitation reaction:

$$CuSO_4 + Fe \longrightarrow Cu + FeSO_4$$

In common with other cement copper operations in the industry, the precipitation is carried out at ambient temperature without pH control. The retention time for the solution in the launders is a few hours. Continuous or batch operation may be used; most plants have continuous operations. Uniform distribution of the pregnant copper solution through the scrap iron is very important in producing a high-grade cement copper. Precipitation is shown in two flow diagrams, Figure 6 and Figure 8.

The plant uses filtration followed by drainage and natural air-drying on a concrete pad to dewater the copper cement which is flushed from the launders. Other facilities conduct the drying operations using a drainage pad or a gas-fired hot plate. During the drying period, the copper precipitate is normally moved as little as possible to minimize the amount of copper oxidation. The moisture content of the dried copper cement is usually 15 to 20 percent.

In the precipitation plant, approximately 0.1 percent (20,000 MT/year) of the tailwater from the precipitation plant is discarded to an evaporation pond as a means of bleeding iron salts out of the leaching circuit. The remaining tailwater is recirculated to the leach dump. The solids in the bleed streams are contained in the evaporation ponds and are the only land-disposed wastes produced from the precipitation of cement copper. No analytical data are available for the solids in the bleed stream.

Solvent Extraction and Stripping Operations.[9,10] The liquid ion exchange process consists of contacting the clarified metal-bearing aqueous solutions with a water-immiscible, kerosene-soluble organic extractant. The flow diagram describing the solvent extraction process is shown in Figure 5. The organic extractant is in a kerosene solution. Copper along with some impurities concentrates in the organic phase, which is subsequently separated from the aqueous phase as the result of immiscibility. The metal-bearing organic phase is then contacted with a new sulfuric acid solution to remove the copper from the organic extractant, resulting in a copper sulfate concentrate.

The chemical reaction taking place in the extraction process can be expressed by the following equation, using the letter R to represent the organic extractant: $CuSO_4 + 2HR = CuR_2 + H_2SO_4$. This reaction shows that in aqueous solutions of copper sulfate at low acid concentrations, the organic will combine with the copper ions to form a loaded organic and will regenerate the sulfuric acid which was associated with the copper. There are three contact stages, each consisting of a mixer and settler. Spent acid solution from the extraction step is sent to heap leaching. The purpose of solvent extraction is to obtain a purified copper sulfate solution suitable for use in electrolysis.

The loaded organic is stripped of copper by treatment with strong sulfuric acid solution, and stripped organic is recycled to the extraction step. The chemical reaction involved in stripping is the reverse of the extraction reaction and is expressed by the following equation: $CuR_2 + H_2SO_4 = CuSO_4 + 2HR$. This reaction demonstrates that when loaded organic is contacted with a high acid strength aqueous solution, the copper is stripped from its organic form into an aqueous copper sulfate solution, and the organic is returned to its hydrogen form, ready to be recontacted with, and extract copper from, slightly acidic aqueous leach liquors.

No significant land-disposed wastes are produced in solvent extraction and stripping operations. All of the liquid streams are recycled to the process.

Electrowinning.[9,10] Electrowinning is carried out in electrolysis cells using purchased starter cathode sheets. Each cell receives pregnant electrolyte at the same rate. The cell solutions (spent electrolyte) overflowing at the opposite end from the feed stream are combined and returned to the stripping operation. Flow diagrams for electrowinning are shown in Figures 5 and 7.

Each cell contains 40 cathodes and 41 anodes. The anodes are 6 percent antimonial lead, and are spaced on 11.4-cm (4.5-in.) centers, with all anodes in any one cell resting on the same bus bar in order to be in parallel electrically. Similarly, all the cathode support bars in that same cell rest on the opposite bus bar. The power consumption is about 0.55 kw-hr/kg of copper.

Copper is deposited on the starter cathodes. After being pulled from the cells, the finished cathodes are soaked in hot water dip tanks and then banded in bundles of 20 cathodes for shipment.

Typical metallurgical data for electrowinning are shown in Figure 5. The product cathodes contain 99.9+ percent copper.

No waste streams of any significant quantity or potential pollution hazard are involved in the electrowinning operations. All liquid streams are recycled in the process.

Identification of Potentially Hazardous Materials. Table 21, p.41, shows the total and potentially hazardous wastes from the mining and concentrating of copper ores. In 1974, there were 140,179,000 MT (154,520,896 tons) dry weight and 461,242,000 MT (508,432,269 tons) wet weight of potentially hazardous wastes resulting from concentrating copper ores.

Tables 22 and 23, p.42 and 43 respectively, show the 1977 and 1983 projections for total and potentially hazardous wastes resulting from the mining and concentrating of copper ore.

Waste Treatment and Disposal

Mining Waste Treatment and Disposal. There are no potentially hazardous wastes destined for land disposal from either underground or open-pit mining of copper ores. The waste rock and overburden that are disposed of on land are either at or below the background level of the minerals present in the mine and the adjacent area. The background concentration data for areas adjacent to the mines were obtained by the U.S. Geological Survey and the University of Idaho.[9]

Concentrating Process Wastes. There are several different concentrating processes used to recover copper from the mined copper ore. Table 24, p.49, is a presentation of the different concentrating processes. The table also shows whether or not waste is generated (i.e., flotation tailings or iron salts left behind in an evaporation pond), whether or not pyrite (FeS_2) is present in the ore, and also indicates if the waste is a potential hazard. Table 19, p.39, shows the number of mines using each process, the total ore mined and copper and waste produced from each process, and the amount of potentially hazardous waste generated.

Flotation. In Table 24, Process Code A shows that the ore is a sulfide, with pyrite present; the process is flotation; the waste material is tailings; and that pyrite is a potential hazard. There are 23 mines mining 148,406,000 MT (163,247,000 tons), producing 1,070,224 MT (1,177,000 tons) of copper, and 140,179,000 MT (154,200,000 tons) of tailings. Because pyrite is present in all the mines in this group, the tailings are considered potentially hazardous, and they constitute 64 percent of the total concentrator wastes from copper recovery processes.

A tailings dam contains concentrator waste which may be potentially hazardous material generated from flotation; therefore, it is important that the dam be constructed to prevent rupture.

Table 26, p.51, shows analytical data on tailings solids reported by all companies contacted by personal visit and through the American Mining Congress questionnaire. As can be seen, very little information was available, and very little interpretation can be made from these data.

The flotation wastes from the 23 mines are disposed of in tailings ponds. There are differences from company to company in the preparation of a tailings pond, in the method of addition of the tailings slurry to the pond, and in the methods used to stabilize the dams and pond surface to assure retention of the contaminants in the tailings pond.

Some companies remove all vegetation from the surface of the area to be used as a tailings pond, and a few companies compact the surface or add material to make the surface impermeable to water. In some copper mining areas of the country, clearing of the vegetation and sealing the surface of the ground is unnecessary, as the tailings ponds are located on top of unvegetated impermeable soil or rock.

The most common method of addition of slurry to the tailings pond is to pump the slurry to the pond, discharging it to the surface of the pond some distance away from the dikes. A few companies add the slurry to the inside face of the dam and utilize the differences in settleability of the fines and coarse to build up the face of the dam with coarser sand. A few companies separate the coarser sand from the fines by using a cyclone, discharging the fines on the inside face of the dike, which aids in sealing the faces of the dikes and dams against leakage. The coarse material is discharged on top of the dam and utilized in raising its level.

Stabilization methods used for the dam and pond surface vary from just keeping water in the pond to extensive measures for stabilization. A few companies have built backup dams downstream from the tailings pond to collect seepage and pump it back into the pond. Several companies are vegetating the surface of the dam and putting in erosion-protection furrows.

One company is compacting the dam continuously and extensively vegetating the face of the dam and the surface of the dam and pond when it is no longer in use.

Some companies that operate underground copper mines also operate sand plants. In these plants, the fines are separated from the coarse material and are pumped to the tailings pond. The coarse material is pumped to the mine where it is mixed with concrete and used to backfill the mine.

The filling of depleted mine stopes with tailings sands from the concentrator waste stream is a well-developed system at these mines. Lumber cut to size in the mine is used to prepare skeleton floor and wall-pouring forms. Wire fencing, burlap, and plastic sheets are used to cover the wooden forms and serve as a seal during the actual sand-filling operation. The tailings sand used as fill material is mixed with cement at the mine surface and pumped down to the fill areas. The bottom 1.22 m (4 ft) of fill consists of 12:1 ratio of sand-to-cement and the remainder has a ratio of 20:1 sand-to-cement. As much water as possible is drained from the sand fill before it sets, in order to obtain a high density and a strong fill. The poured fill sets up in about 24 hr. After the fill has set, the adjacent pillars are mined out to improve the ore recovery from the mine operations.

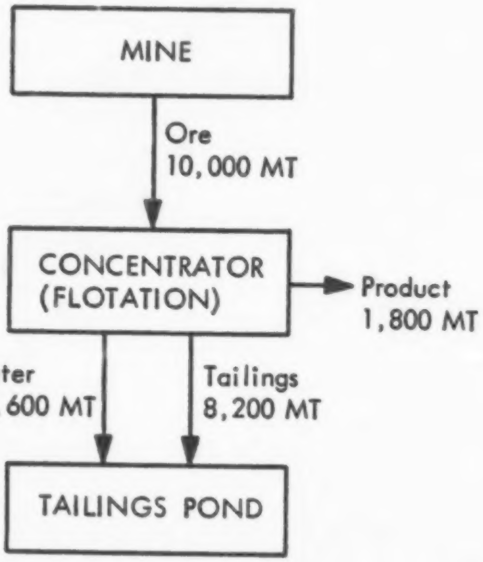
Leaching. In leaching of ore with sulfuric acid, there is no potentially hazardous waste generated.

Electrowinning. There is no potentially hazardous waste generated in electrowinning of copper.

Levels I, II, and III Technology. Level I technology for disposal of potentially hazardous wastes from copper flotation plants is the use of tailings ponds; all flotation concentrators use tailings ponds. Figure 9 shows the waste disposal method for Level I technology used by all concentrators. The potentially hazardous wastes, dry solids, amount to 82 percent of the ore processed in the concentrator. This material is disposed on land by pumping the solids and water to a tailings pond. The slurry pumped to the pond is roughly 31 percent solids. Level II technology for disposal of potentially hazardous wastes is the vegetative stabilization of the dams and dikes and attention to surface preparation of the area before establishment of a tailings pond, plus the compaction of the dam and dike to prevent erosion, and treatment with lime or limestone to neutralize the effect of the pyrite. Figure 10 shows the waste disposal method for Level II technology. This method is presently used by six mines. The material balance for Figure 10 indicates that 41 percent of the ore processed is pumped to the tailings pond with the water used in the flotation process. Another 41 percent of the potentially hazardous waste material is pumped back to the underground mine, mixed with cement and used to backfill the mine stopes. The slurry pumped to the tailings pond is about 33 percent solids and contains the fine, smaller than 65 mesh, material.

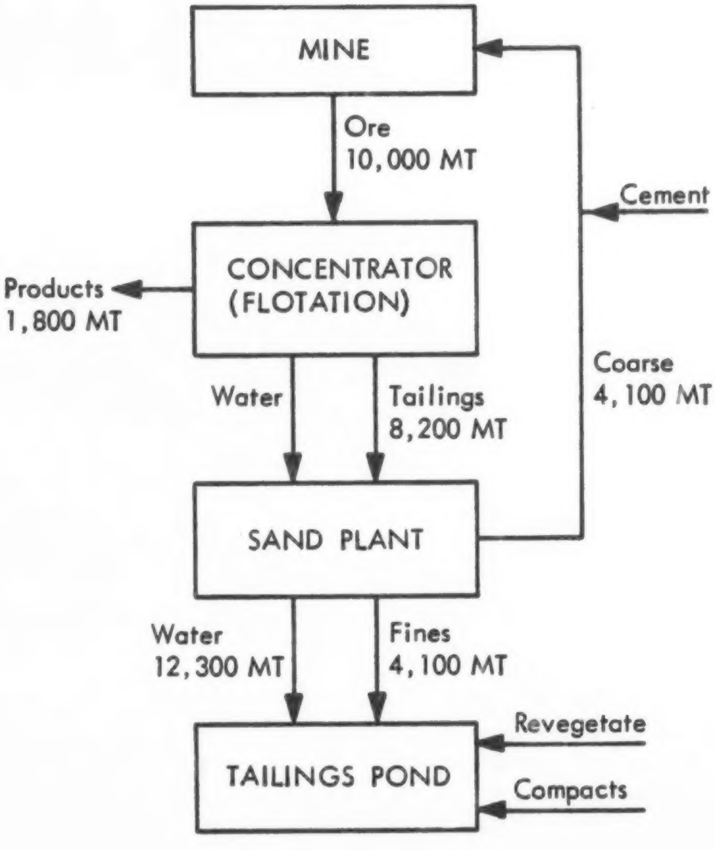
Level III technology for safe disposal is to incorporate effluent wastewater treatment. Two of the largest mines are treating the effluent from the pond in a wastewater treatment plant (and returning the solids precipitated from the water to the tailings pond) before discharging the effluent. This practice reduces or eliminates the effect of acid formation due to pyrite and subsequent leaching of metals in the tailings. Figure 11 shows the waste disposal techniques for Level III technology. The solids being returned from the wastewater plant cannot be quantified at this time. The wastewater treatment plants did not operate until the last quarter of 1974, and solids were not returned to the tailings pond during 1974.

The enforcement of air and water pollution regulations for 1977 and 1983 will have only a very small effect, less than 1 percent, on the tonnage of potentially hazardous wastes disposed on land from copper concentrating operations. There are very few operations in a copper flotation plant that generate an air pollutant. If dry grinding of ore is subject to air regulation, wet grinding of the ore is an obvious solution. Most concentrators wet grind at the present time.


MINE
Ore
10,000 MT
CONCENTRATOR
(FLOTATION)
Product
1,800 MT
Water
26,600 MT
Tailings
8,200 MT
TAILINGS POND

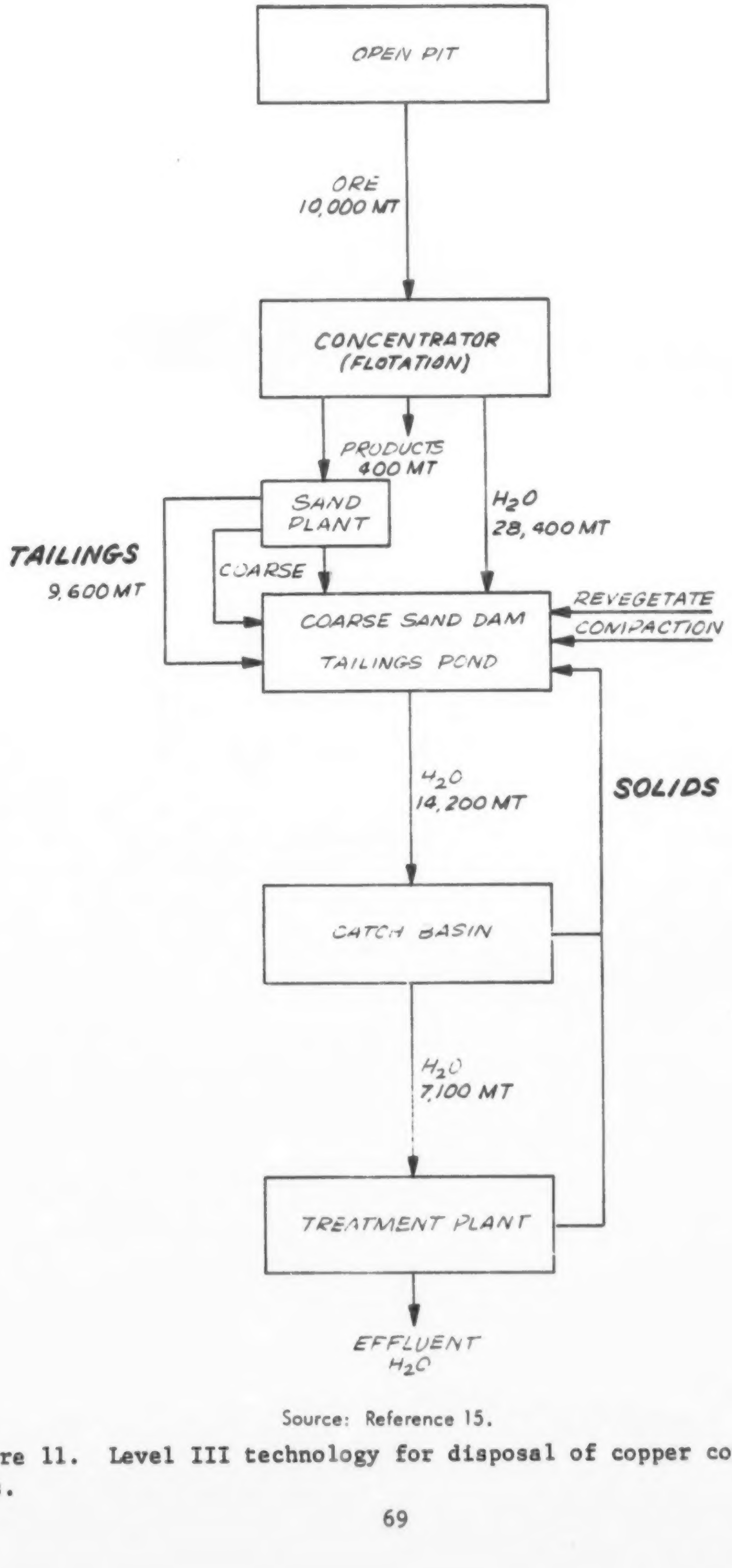
Source: Reference 15.

Figure 9. Level I technology for disposal of copper concentrator wastes.


MINE
Ore
10,000 MT
Cement
CONCENTRATOR
(FLOTATION)
Products
1,800 MT
Coarse
4,100 MT
Water
Tailings
8,200 MT
SAND PLANT
Water
12,300 MT
Fines
4,100 MT
Revegetate
TAILINGS POND
Compacts

Source: Reference 15.

Figure 10. Level II technology for disposal of copper concentrator wastes.


OPEN PIT
ORE
10,000 MT
CONCENTRATOR
(FLOTATION)
PRODUCTS
400 MT
SAND
PLANT
TAILINGS
9,600 MT
COARSE
H_2O
28,400 MT
COARSE SAND DAM
TAILINGS POND
REVEGETATE
COMPACTION
H_2O
14,200 MT
SOLIDS
CATCH BASIN
H_2O
7,100 MT
TREATMENT PLANT
EFFLUENT
H_2O

Source: Reference 15.

Figure 11. Level III technology for disposal of copper concentrator wastes.

Effluent limitations arising from enforcement of the guidelines will require removal of trace metals from the effluent water. These solids will add to the potentially hazardous waste destined for land disposal, but should contribute < 0.1 percent of the total wastes in 1977 and 1983. Zero discharge will require that all concentrators install wastewater treatment to remove traces of metal ions now present in the discharge. New technology will have to be developed to remove all cations from the effluent.

All technology identified in the program can be transferred to any copper mine where it is necessary to properly dispose of potentially hazardous waste materials for the protection of the environment. Delivery of equipment and construction required to enable a concentrator to move from Level I to Level III technology could require 3 to 5 years.

The application of Level III technology to disposal of flotation tailings waste where pyrite is present would mean the consumption of considerable energy and some investment in equipment. There are about 18 concentrating plants that will have to install wastewater treatment facilities, pumps to deliver the effluent to the treatment plant, and solids handling equipment to transfer the treatment plant sludges back to the tailings pond.

Table 18 shows the total wastes disposed on land by copper concentrators. There were 240,794,000 MT dry weight (265,430,000 tons) of solids disposed of on land in 1974. There are no radioactive materials in the copper wastes. The copper and other trace materials present in the tailings at certain concentration levels are potentially hazardous and must be kept in the tailings pond. The material in the pond is about 60 to 80 percent -60 mesh and has the physical appearance of sand. There is no reliable information available on the chemical composition. Most tailings will contain from 100 to 1,500 parts per million of copper. Table 26 contains all of the analytical information furnished by 15 mines on copper concentrator tailings.

The bulk of the material in the tailings is not soluble in water and most other solvents, including acids. Copper is insoluble until the pH of the contacting solution is below 3.5. All metals found in the tailings have a very limited solubility between 3.5 and 7 pH, and the solublity above 8 pH is below the recommended drinking water standard. If the solutions coming in contact with the tailings pond are above 7 pH, very little hazard exists.

There are no intermedia pollution problems where Level II or Level III technology is implemented. The only problem is in tailings ponds where a considerable quantity of pyrite is present. In this case, sulfuric acid could be made by water and air contact with the FeS_2 (pyrite), and the sulfuric acid would leach some metals from the tailings, unless lime or other caustic material is added in the flotation process.

The only places where sampling is needed is where pyrite is present at concentrations > 2 percent in the ore and is not neutralized by lime addition. Samples of the effluent water should be taken monthly and analyzed for pH and metals. Stabilization techniques and progress toward stabilization should be monitored by quarterly visual surveillance of the tailings pond.

Waste Disposal Costs - Copper

Waste Disposal Costs, Tailings Pond Using Level I Technology. The representative copper mine used in the cost analysis produces 834,610 MT (918,000 tons) of ore per year. The representative plant chosen for the cost estimate is a model plant, based on average figures and practices of several copper mines. A material balance was calculated for a number of concentrators, all using the same process, flotation, and all using a tailings pond for disposal of their potentially hazardous waste. An average ore grade of 1 percent was used in calculating the material balance. A concentrate with 25 percent copper was also used. After the material balance data were derived, a model plant was designed and used as the representative plant. A model plant was used to ensure that a particular mine and concentrator could not be identified. After the concentrating process, 442,706 MT (487,000 tons) of dry tailings per year are deposited in the tailings pond along with 4,470,000 MT (4,920,000 tons) of water per year. The tailings pond covers 12.1 hectares (30 acres) and is located within 0.8 km (0.5 miles) of the concentrator. The major assumptions of the analysis are presented in Table 27. Many of the costs are similar to those associated with the uranium tailings pond.

The results of the analysis are presented in Table 28. Two cost estimates are presented. The differences arise due to different assumptions about the value of the land after 20 years. The costs per metric ton of disposal of these potentially hazardous wastes are estimated to be between \$0.275 and \$0.282/MT (\$0.25-\$0.26/ton). Costs estimated by a copper mining corporation using a tailings pond of approximately the same size as the representative plant equal \$0.38/MT (\$0.35/ton). If the entire copper mining and concentrating industry utilized technology Level I, the cost would be 6.6 to 6.8 million dollars per year. This cost represents less than 1 percent of the value added by the copper mining industry.

Waste Disposal Costs, Tailings Pond Using Level II Technology. The disposal of potentially hazardous wastes from copper concentrators is more expensive using Level II than Level I technology. In addition to the tailings pond, Level II technology also requires:

TABLE 27

MAJOR COST ASSUMPTIONS, POTENTIALLY HAZARDOUS WASTE FROM COPPER CONCENTRATORS -- TECHNOLOGY LEVEL I

I. Capital

A. Land

12.1 ha (30 acres) are required for pond*
Land cost at $12,350/ha ($5,000/acre) = $150,000
Levelized annual land cost (capital recovery factor of 0.117) = $17,550/year

B. Tailings Pump and Pipeline

Tailings pond assumed to be within 0.8 km (0.5 mile) of concentrator*
Tailings pump and pipeline cost = $393,016†
Levelized annual cost at capital recovery factor of 0.117 = $45,982/year

C. Earth Dam

Earth dam capital cost = $35,000
Levelized annual cost = $4,095/year

D. Cyclone Installation

Capital cost of cyclone = $58,000
Levelized annual cost = $6,786/year

II. Labor

1 operator required for cyclone separator for 5 months, 8 hr/day at $8.97/hr§
Labor cost = $7,176/year

III. Supervision

Assumed to be 25% of labor cost§
Supervision = $1,794/year

IV. Maintenance

Assumed to equal 6% of nominal cost of equipment/year§
Maintenance cost = $27,061/year

TABLE 27 (CONCLUDED)

V. Insurance and Taxes

Assumed to equal 2%/year of total nominal capital cost (land and equipment)§
Insurance and taxes = $12,720/year

VI. Energy and Power

A. Tailings pump is assumed to consume electricity with a value of approximately $1,400/year**
B. Cyclone separator mounted on truck--only fuel required is for truck as it travels around the pond. Assumed to be less than $100/year

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* MRI communication from individual copper mining corporation.

† Sears, M. B., et al. Correlation of radioactive waste treatment costs and the environmental impact of waste effluents in the nuclear fuel cycle for use in establishing as low as practicable guides - Milling of uranium ores. Oak Ridge National Laboratory, Oak Ridge, Tennessee, ORNL-TM-4903, v.1, May 1975. p. 187-188. Obtained by ratio of material handled (dry millings and water) to total cost for pump and pipeline.

§ See corresponding table for uranium tailings pond.

** Obtained in the same manner as the capital cost of the tailings pump and pipeline (see footnote †).

TABLE 28

DISPOSAL COSTS, POTENTIALLY HAZARDOUS WASTE FROM COPPER CONCENTRATORS, TAILINGS POND, TECHNOLOGY LEVEL I (EXPRESSED IN EQUIVALENT ANNUAL 1973 DOLLARS)

	Case "A"*	Case "B"†
Capital (levelized cost)		
Land	17,550	14,672
Other capital	56,863	56,863
Labor	7,176	7,176
Supervision	1,794	1,794
Maintenance	27,061	27,061
Insurance and taxes	12,720	12,720
Energy and power	1,500	1,500
Total	124,664	121,786
Metric tons of potentially hazardous waste deposited in pond annually	442,706	442,706
Cost per metric ton of potentially hazardous waste	$0.282	$0.275
Total cost if entire industry uses the technology level ($\$10^6$/year)	$6.8	$6.6
Total cost as a percent of value added in mining‡	0.6%	0.6%

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* "A" assumes all land is purchased in Year 1 and has 0 resale value after 20 years.

† "B" assumes all land is purchased in Year 1 and is resold at the same value after 20 years.

‡ 1972 Census of Mineral Industries, Bureau of Census, Government Printing Office, 1975 corrected for inflation to 1973.

- Vegetative stabilization of dams and dikes;
- Attention to surface preparation before tailings pond is built;
- Compaction of dams and dikes to prevent erosion;
- Treatment with lime or limestone.

Assumptions concerning each of these additional cost factors are presented in Table 29. Table 30 presents the results of the analysis. Vegetative costs per hectare are substantial. However, when the cost is distributed over 20 years of analysis, the annual equivalent costs are small. Surface preparation and compaction costs can also be significant, even on the 12.1-ha (30-acre) tailings pond. Lime treatment costs, on the other hand, are part of the concentration process. No cost has been calculated for this treatment, because it is not done for disposal purposes. The results indicate that Level II technology disposal costs are approximately $0.32 to $0.33/MT of potentially hazardous wastes. Total industry costs, assuming all companies utilize Technology Level II, would be 77.1 to 79.4 million dollars per year. This cost would amount to 7.2 percent of the value added in mining and concentrating copper ores.

Waste Disposal Costs, Tailings Pond and Wastewater Treatment Plant Using Level III Technology. Level III technology differs from Level II waste disposal techniques in that a wastewater treatment plant is utilized. Effluent from the tailings pond is treated in the plant. Solids precipitated from the water are returned to the tailings pond by truck, and the treated water is discharged.

The additional assumptions used in calculating the Level III technology costs are presented in Table 31. Capital, operation, maintenance, materials, and other costs are presented in the table. The results of the analysis are presented in Table 32. All costs incurred in Level II technology are also present in Level III, in addition to the wastewater treatment costs. Table 32 indicates that the disposal costs of potentially hazardous wastes range from $0.522 to $0.516/MT ($0.475-$0.469/ton) of waste. Total industry costs would range from 125.7 to 124.3 million dollars per year or 11.4 percent to 11.3 percent of the total value added in mining and concentrating copper ores.

TABLE 29

ADDITIONAL COST ASSUMPTIONS--TAILINGS POND--COPPER CONCENTRATORS--TECHNOLOGY LEVEL II

I. Vegetative Stabilization of Dams and Dikes

1.05 ha (2.6 acres) of berm are assumed to require vegetative stabilization*

Cost of vegetative stabilization assumed to be \$3,213/ha (\$1,298/acre)*

Vegetative stabilization cost = \$3,375

Levelized cost per year = \$395/year

II. Surface Preparation Before Pond Construction

12.1 ha (30 acres) of pond area are assumed to be bulldozed and compacted

Time required: approximately 16 hr of bulldozing and compacting at 3 mph†

Total cost of surface preparation:

1. Labor at \$8.97/hr‡ = \$142.52

Levelized equivalent annual labor cost = \$17/year

2. Energy and power (60 gal/8 hr, at \$0.50/gal)‡ = \$60.00

Levelized equivalent annual cost = \$7.00/year

III. Compaction of Dams and Dikes to Prevent Erosion

A. Capital

One compactor at \$70,000 required with useful life of 10 years§

Present value of compactor capital costs at 10% discount rate = \$97,020

Equivalent annual cost = \$11,351/year

B. Labor

Compactor assumed to be used 8 hr/week§

Annual labor cost at \$8.97/hr = \$3,732/year

TABLE 29 (CONCLUDED)

C. Maintenance

Assumed to equal 6% of nominal capital cost or $4,200/year

D. Energy and Power

Compactor assumed to consume 3 gal/hr at $0.50/gal*
Annual energy plus power costs = $624/year

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[*] Ludeke, K. Vegetative stabilization of tailings disposal berms. *Mining Congress Journal*, January 1973. p. 32-39. Berm area obtained from ratio of berm requiring vegetation to tailings pond size obtained from Pima Copper Mine.

[†] MRI estimate for 30 acre rectangular tailings pond.

[‡] See previous discussion of uranium open-pit mine disposal costs.

[§] MRI estimate.

TABLE 30

DISPOSAL COSTS, POTENTIALLY HAZARDOUS WASTE--COPPER INDUSTRY--TAILINGS POND, TECHNOLOGY LEVEL II (EXPRESSED IN EQUIVALENT ANNUAL 1973 DOLLARS)

	Case "A"*	Case "B"†
Capital	85,764	82,886
Labor	10,925	10,925
Supervision	2,732	2,732
Maintenance	31,261	31,261
Insurance and taxes	12,720	12,720
Energy and power	2,131	2,131
Vegetation of dams and dikes	395	395
Total	145,928	143,050
Metric tons of potentially hazardous tailings deposited in pond annually	442,706	442,706
Cost per metric ton of potentially hazardous waste to tailings pond	\$0.330	\$0.323
Total cost if entire industry used the technology level (\$$10^6$/year)	\$79.4	\$77.1
Total cost as a percent of value added in mining	7.2%	7.2%

* Case "A" assumes all land is purchased in year 1 and has 0 value after 20 years.

† Case "B" assumes all land is purchased in year 1 and is resold at the same value after 20 years.

TABLE 31

ADDITIONAL COST ASSUMPTIONS--COPPER TAILINGS POND--TECHNOLOGY LEVEL III

Wastewater Treatment Plant

A. Capital

4,470,000 MT/year of water enter the pond
50% evaporation assumed, 25% of water assumed to be held in solids and 25% assumed treated as effluent*
Therefore, 809,000 gal/day will enter plant
Plant contains two systems: agitated open tank and sedimentation system
1. Agitated tank capital cost = $90,000 (16,860 gal. capacity)†
Useful life = 20 years‡
2. Sedimentation system capital cost = $50,000 (809,000 gal/day)§
Useful life = 40 years§
Using S.L. depreciation, net capital cost = $46,900
Equivalent annual cost (tank and sedimentation) = $16,017/year

B. Operation and Maintenance (includes energy and power)

1. Agitated tank operation labor costs, 4,380 hr/year at $8.97/hr = $40,000‡
Agitated tank maintenance costs = $9,000/year
2. Sedimentation system operation costs are minimal§
Sedimentation system maintenance costs = $5,250/year§

C. Materials

60 kg of dolomitic lime required per day**
Lime cost equal $22/MT††
Annual lime cost = $482/year

D. Other cost

Assumed one load of sludge per day is trucked back to tailings pond
Labor cost = $3,274/year

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* The amount of water treated can vary from 10 to 25 percent of water entering the pond.

† Blecker, H., and T. Nichols. Capital and operating costs of pollution control equipment modules. Environmental Protection Agency, v.II, EPA-R5-73-0236, July 1973. p. 10.

‡ Blecker, H., and T. Nichols. Capital and operating costs of pollution control equipment modules. Environmental Protection Agency, v.II, EPA-R5-73-0236, July 1973. p. 11.

§ Blecker, H., and T. Nichols. Capital and operating costs of pollution control equipment modules. Environmental Protection Agency, v.II, EPA-R5-73-0236, July 1973. p. 126-127.

** MRI estimate.

†† Environmental Protection Agency. Development document for effluent limitation guidelines and new source performance standards for the major inorganic products. EPA-440/1-74-007-a, March 1974. p. 275.

TABLE 32

DISPOSAL COSTS, POTENTIALLY HAZARDOUS WASTE--COPPER CONCENTRATOR TAILINGS POND AND WASTEWATER TREATMENT PLANT, TECHNOLOGY LEVEL III
(EXPRESSED IN EQUIVALENT ANNUAL 1973 DOLLARS)

	Case "A"*	Case "B"†
Capital		
Wastewater treatment	16,017	16,017
Other capital	85,764	82,886
Labor	54,199	54,199
Supervision	13,550	13,550
Maintenance	45,511	45,511
Insurance and taxes	13,040	13,040
Energy and power	2,131	2,131
Vegetation of dams and dikes	395	395
Materials	482	482
Total	231,089	228,211
Metric tons of potentially hazardous tailings deposited in pond annually	442,706	442,706
Cost per metric ton of potentially hazardous waste	\$0.522	\$0.516
Total cost if entire industry utilized the technology level (\$$10^6$/year)	\$125.7	\$124.3
Total cost as a percent of value added in mining	11.4%	11.3%

* Case "A" assumes all land is purchased in Year 1 and has 0 resale value after 20 years.

† Case "B" assumes all land is purchased in Year 1 and is resold at the same value after 20 years.

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11. Hannan, W. S., Jr. Leach dump operations at Bisbee. Phelps Dodge Corporation, Copper Queen Branch, Bisbee, Arizona, Oct. 5, 1971.

12. Miller, A. Process for the recovery of copper from oxide copper bearing ores by leach liquid exchange and electrowinning at Ranchers Bluebird Mine. Ranchers Exploration and Development Corporation.

13. Ward, M. H. Engineering for in situ leaching Ranchers Exploration and Development Corporation. Miami, Arizona, Jan. 1973.

14. Jacky, H. W. Copper precipitation methods at Weed Heights. Mining Engineering, Society of Mining Engineers, June 1967.

15. MRI communications. Copper mining companies.

SECTION II

LEAD-ZINC ORES (SIC 1031)

Industry Characterization

History of the Industry. Lead is one of the oldest metals used by man, and many of the ancient applications have persisted through the centuries. The earliest known specimen of lead, dating from 3000 B.C., is a figure found in the area of the Dardanelles on the site of the ancient city called Abydos. Old lead pipes have been found in Egypt, and the hanging gardens of Babylon were floored with soldered sheets of lead.

The lead mines of Cyprus, Sardinia, and Spain were worked by the Phoenicians, and the lead-silver mines of Laurium, Greece, are famed as the source of ancient treasures. The great quantities of silver that assisted in the rise of Rome were derived from lead smelting-refining in Britain, Sardinia, and Spain.

In the United States, lead was mined and smelted in Virginia as early as 1621, and the discovery of lead in the Upper Mississippi Valley was reported in 1690. These shallow, easily processed ores were important sources of domestic lead during much of the 19th century. The production averaged about 907 MT (999 tons) annually from 1801 through 1810 and increased to about 2,721 MT (2,999 tons) per year from 1831 to 1840, some 10 percent of the world output. In 1867, discoveries in southeast Missouri led to development of one of the most productive lead-zinc areas in the world. Since 1904, southeastern Missouri has been the leading producing lead-zinc district in the United States.

Lead is easily worked and noncorrosive, and since medieval times has been used in piping, building materials, solders, paints, type, ammunition, and castings. The advent of the electric starter on internal combustion engines was followed by a large increase in consumption, which was expanded by the need for antiknock gasoline additives for use in high compression engines. United States consumption of lead first reached 1 million metric tons (1,102,311 tons) in 1941 and has been well above that figure in most years since. Lead consumption amounted to nearly 1.5 million metric tons (1,653,466 tons) in 1973.

In most lead mines the ore is composed of a combination of lead and zinc sulfides and the concentrators recover both lead and zinc concentrates.

Today, the United States is the world's leading producer of lead and also the leading consumer. Lead mining and refining is a major basic industry and ranks fifth worldwide, following steel, aluminum, copper, and zinc in tonnage of metal produced. Some 53 countries, well distributed throughout the world, produce lead. In 1973, the leading producers were the United States, U.S.S.R., Australia, Canada, Mexico, Peru, and Yugoslavia.

Domestic Production and Capacities. During the period 1963 to 1973, United States lead production increased from 229,000 MT (252,429 tons) to 548,000 MT (604,067 tons) (recoverable content of ore)* (Table 33). This increase was the largest in the world during the 11-year period, and the United States is the leading world producer. While Government statistics are not available for 1974, it is estimated that United States production of lead concentrate product was 1,231,652 MT† (1,357,664 tons) (Table 34). Table 35 gives the ore produced by year, while Tables 36 and 37 give the mine production of recoverable lead by state and region, respectively.

TABLE 33

LEAD PRODUCTION, 1963-1983 (1,000 MT)*

Year	United States	World total
1963	229	2,544
1964	259	2,481
1965	273	2,691
1966	296	2,847
1967	287	2,865
1968	325	2,992
1969	461	3,196
1970	518	3,392
1971	524	3,403
1972	561	3,491
1973	548	3,445
1977†	585	--
1983†	643	--

Source: References 1 and 2.

* Data published in tons and calculated to metric tons.

† Estimated production.

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* Recoverable content of ore = the amount of metal produced after the recovery processes have been completed.

† Recovered lead concentrate product = the amount of concentrate produced before smelting and refining the ore.

TABLE 34

U.S. LEAD CONCENTRATE PRODUCTION FOR 1974 (RECOVERED LEAD CONCENTRATE PRODUCTS)

State	Metric tons
California	20,629
Colorado	69,887
Idaho	150,868
Illinois	57,930
Kentucky	2,268
Missouri	638,897
Montana	6,668
New Mexico	6,000
New York	226,796
Utah	31,751
Virginia	3,629
Washington	1,814
Wisconsin	14,515
U.S. total	1,231,652
EPA regions	
Region I	--
Region II	226,796
Region III	3,629
Region IV	2,268
Region V	72,445
Region VI	6,000
Region VII	638,897
Region VIII	108,306
Region IX	20,629
Region X	152,682

Source: Reference 3.

TABLE 35

TOTAL LEAD MINE PRODUCTION OF ORE BY YEAR
(ORE PRODUCED)

Year	1,000 MT
1967	287,514
1968	325,820
1969	461,768
1970	518,698
1971	524,851
1972	561,470
1973	547,054

Source: References 4 and 5.

TABLE 36

MINE PRODUCTION OF RECOVERABLE LEAD IN THE UNITED STATES, BY STATE (MT)

State	1969	1970	1971	1972	1973
Alaska	2	--	--	--	5
Arizona	197	259	779	1,599	692
California	2,284	1,608	2,072	1,046	40
Colorado	19,747	19,827	23,356	28,437	25,503
Idaho	59,509	55,530	60,428	55,707	56,013
Illinois	718	1,390	1,123	1,211	491
Kansas	358	73	--	--	--
Maine	--	--	--	77	185
Missouri	322,461	382,618	389,757	443,973	441,929
Montana	1,590	904	558	260	160
Nevada	1,288	330	101	*	--
New Mexico	2,148	3,221	2,695	3,250	2,319
New York	1,530	1,161	796	988	2,090
Oklahoma	549	723	--	--	--
Oregon	*	*	--	--	--
South Dakota	1	3	--	--	--
Utah	37,496	41,165	34,718	18,784	12,458
Virginia	3,046	3,045	3,072	3,122	2,392
Washington	7,846	6,154	4,696	2,329	2,011
Wisconsin	1,000	690	682	687	766
Other states	--	--	18	--	--
Total	461,768	518,698	524,851	561,470	547,054

Source: Reference 4.

* Less than one-half unit.

TABLE 37

MINE PRODUCTION OF LEAD IN THE UNITED STATES, BY EPA REGION FOR 1973

Region	MT
I	185
II	2,090
III	2,392
IV	--
V	1,257
VI	2,319
VII	441,929
VIII	38,121
IX	732
X	58,029

Source: Reference 4.

The available supply of lead has exceeded U.S. industrial demand, resulting in a buildup in stocks. The 3 percent average annual growth in demand since 1965 has been due primarily to the growth of transportation requirements for batteries. This growth in transportation end product applications has been in contrast to the relatively stable tonnage or even declining consumption for many historical end products used in construction and communications. The Environmental Protection Agency's regulations reducing the lead content of gasoline by 60 to 65 percent over a 4-year period were scheduled to become effective on January 1, 1975, and will curtail growth in demand for lead.

Domestic demand for lead is forecast to increase at an annual rate of approximately 1.6 percent through 1983. To meet the projected demand, lead production is expected to increase to 585,000 MT (644,852 tons) of metal in 1977 and 643,000 MT (708,786 tons) in 1983. The domestic resource base is more than adequate to support the domestic demand at competitive prices, and reliance on imports probably will decline from the current level of about 30 percent of total primary metal consumption.

Number, Location, Size, and Age of Mines and Concentrators. The domestic lead-zinc mining industry in 1974 comprised 30 mining operations, with about 39 individual mines and 32 concentrators located in 11 states (Table 38). Lead-zinc mines vary in size from small operations employing fewer than 50 workers to mines employing more than 1,000 workers. Recent data from the Engineering & Mining Journal and the Mining Enforcement and Safety Administration indicate that 74 percent of the operations employed more than 100 workers in 1974 (Table 38).

TABLE 38

NUMBER, LOCATION AND SIZE OF ACTIVE LEAD-ZINC MINING AND CONCENTRATING OPERATIONS

State	Employees 1-19	20-49	50-99	100-249	250-499	500-999	1,000-2,499	2,500+	Total operations	Total number of mines	Total number of concentrators
California		1							1	0	1
Colorado			2	2	2				6	6	6
Idaho		1	1	1	1		1		5	6	5
Illinois				2					2	2	2
Kentucky		2							2	2	2
Missouri				4	3				7	10	7
New Mexico		1		1	1				3	3	3
New Jersey				1					1	1	1
New York						1			1	2	1
Utah					1				1	2	1
Virginia					1				1	2	1
Washington			1						1	2	1
Wisconsin			1						1	1	1
U.S. total		5	5	11	9	1	1		32	39	32
EPA regions											
Region I									0	0	0
Region II					1	1			2	3	2
Region III					1				1	2	1
Region IV		2							2	2	2
Region V			1	2					3	3	3
Region VI		1		1	1				3	3	3
Region VII				4	3				7	10	7
Region VIII			2	2	3				7	8	7
Region IX		1							1	0	1
Region X		1	2	1	1		1		6	8	6

Source: References 3 and 6.

The ages of active lead-zinc mines vary considerably; some have been in operation more than 90 years, while others have only recently opened. The Bunker Hill mine in Idaho was opened in 1885, while the Cerro Spar Corporation mine in Kentucky was opened in 1974. The year of mine openings is not known for all active lead-zinc mines, however, data for the 17 mines for which information is available, indicate that active lead-zinc mines are comparatively young in age, with 14 mines opening after 1940 and nine mines since 1960.

In 1972, the 25 largest lead-zinc mines accounted for over 95 percent of domestic production, and the leading seven mines accounted for 74 percent of the total domestic primary production. Currently, domestic lead-zinc mine production is located in 11 states (Figure 12); this figure also shows the zinc mines. Five States--Missouri, Idaho, Colorado, New York, and Utah--produced 73 percent, with Missouri contributing 50 percent of the total recovered lead concentrate products in 1974. The world lead-zinc mining industry operated at an estimated 86 percent of capacity in 1972. In the United States curtailment of production to reduce inventories, as well as a shortage of skilled underground personnel, held output about 3 percent below capacity (Tables 39 and 40). A major increase in lead-zinc mine capacity was affected during the last 5 years. The increases are mainly expansion of existing mines, although one new mine in Missouri will offset reduction in output at presently operating mines.

Employment. Total estimated employment in the 38 active lead-zinc mines in 1974 was approximately 6,100 according to figures issued by Engineering & Mining Journal, Mining Enforcement and Safety Administration, and from on-site investigations made by Midwest Research Institute. More than 50 percent of the lead-zinc workers were employed in Missouri, Idaho, and Colorado where the major lead mines are located (Table 41).

By-Products and Coproducts. The domestic mine production of lead-zinc continues to come chiefly from ores mined primarily for their lead and zinc content. Additional lead was derived from ores in which lead and zinc were comparably valued as coproducts and as a coproduct of ores mined for copper, gold, silver, zinc, and fluorspar. The complex ores of the Rocky Mountain area are particularly dependent for economic extraction on the aggregate value of the lead, zinc, silver, and gold content, and distinction as to the leading base metal in value is variable. In addition to lead, the principal metals recovered in processing lead ores and concentrates are antimony, bismuth, gold, silver, tellurium, zinc, and copper. Appreciable quantities of sulfur also are recovered as coproducts of lead. The relationship of the various associated metals and nonmetals to lead production is shown in Table 42.

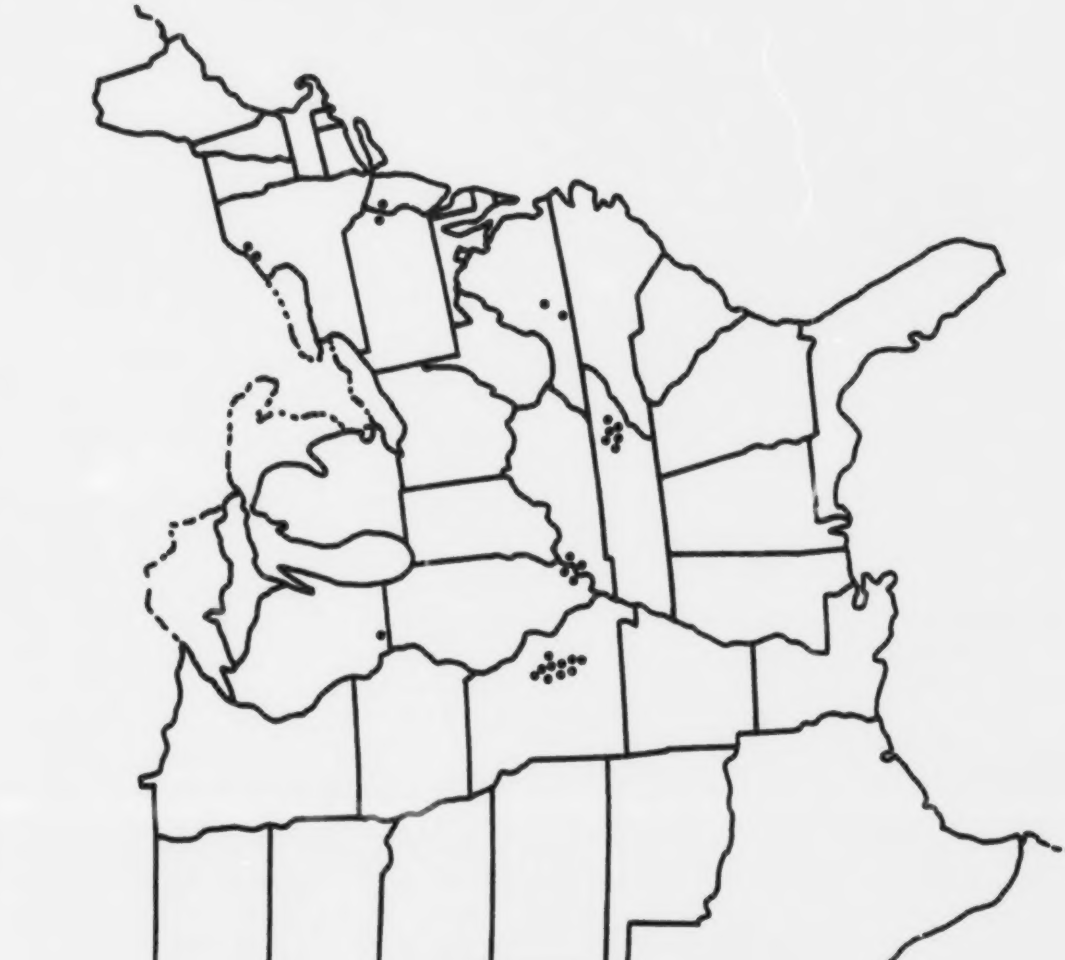


Figure 12. Location of lead-zinc and zinc (SIC 1031) mines.

TABLE 39

U.S. LEAD CAPACITY AND PRODUCTION FOR 1972
(1,000 MT)*

	Capacity	Production (recoverable content of ore)
Mine	580	561
Smelter	712	638
Refinery	712	638

Source: Reference 2.
* Data published in tons and calculated to metric tons.

TABLE 40

PROJECTED U.S. LEAD PRODUCTION CAPACITY FOR 1972-1975
(1,000 MT)

	Production (recoverable content of ore)*			
	1972	1973	1974	1975
Mine	580	589	589	589
Smelter	712	712	712	712
Refinery	712	712	712	712

Source: Reference 2.
* Data published in tons and calculated to metric tons.

TABLE 41

EMPLOYMENT AT ACTIVE LEAD-ZINC MINE AND CONCENTRATOR OPERATIONS

State	Employment
California	48
Colorado	989
Idaho	1,669
Illinois	275
Kentucky	40
Missouri	1,389
New Mexico	569
New Jersey	156
New York	501
Utah	286
Washington	92
Wisconsin	87
U.S. total	6,101
EPA regions	
Region I	--
Region II	657
Region III	--
Region IV	40
Region V	362
Region VI	569
Region VII	1,389
Region VIII	1,275
Region IX	48
Region X	1,761

Source: References 3 and 6.

TABLE 42

DOMESTIC LEAD BY-PRODUCT AND COPRODUCT RELATIONSHIPS (1972)

Ore source	By-product and coproduct	Quantity* (MT)	Percent of total product output
Lead	Bismuth	W	100.0
Lead	Antimony	468	51.4
Lead	Silver	226	20.3
Lead	Zinc	74,000	17.2
Lead	Tellurium	W	W
Lead	Gold	1.5	3.4
Lead	Copper	11,800	0.8
Lead	Sulfur	W	W
Lead	Lead	490,000	87.4
Zinc	Lead	66,000	11.8
Silver	Lead	2,700	0.5
Copper	Lead	900	0.2
Fluorine	Lead	W	W
Gold	Lead	9.9	--

W = Withheld to avoid disclosure.
Source: Reference 2.
* Data calculated to metric tons.

Waste Generation and Characterization

Quantities and Types of Waste Generated. Over 14 million metric tons (15 million tons) of wastes were generated by the lead-zinc ore industry in 1974. These wastes included 1,866,000 MT (2,570,000 tons) of waste rock, and 12,428,000 MT (13,700,000 tons) of tailings.

The amounts of lead-zinc wastes generated in each producing state and EPA region are given in Table 43. The states which produce the most wastes are Missouri followed by Idaho, New York, and Colorado. Missouri, Region VII, produced 42 percent of the total wet wastes.

The ratios of waste rock and concentrator wastes to ore mined are shown in Table 44.

TABLE 43

TOTAL PRODUCTION STATISTICS BY STATE AND EPA REGION FOR
LEAD-ZINC AND ZINC ORES IN 1974 FOR SIC 1031 (METRIC)

				Concentrator tailings		Concentrator products								
State	EPA Region	Ore mined 10^3 TPY	Waste rock 10^3 TPY	Dry 10^3 TPY	Wet 10^3 TPY	Lead conc. 10^3 TPY	Zinc conc. 10^3 TPY	Copper conc. 10^3 TPY	Miscellaneous 10^3 TPY	Total 10^3 TPY	Ratio of dry weight waste to ore	Ratio of dry weight waste to product	% Total dry waste in region	% Total wet waste in region
New Jersey		183	0	123	615	0	59	0	0	59	0.67	2.1		
New York		1,306	5	934	4,670	227	146	0	0	373	0.72	2.5		
	II	1,489	5	1,057	5,285	227	205	0	0	432	0.71	2.3	7.43	7.14
Pennsylvania		544	0	406	2,265	0	54	0	41	95	0.75	4.3		
Virginia		544	12	141	2,091	4	28	0	404	436	0.28	0.35		
	III	1,088	12	547	4,356	4	82	0	445	531	0.51	1.05	3.91	5.89
Kentucky		59	0	0	-	2	2	0	54	58	--	--		
Tennessee		3,184	78	67	9,907	0	156	0	2,961	3,117	0.05	0.05		
	IV	3,243	78	67	9,907	2	158	0	3,015	3,175	0.04	0.05	1.02	13.39
Illinois		294	5	102	510	58	61	0	73	192	0.4	0.56		
Wisconsin		468	0	441	2,205	15	12	0	0	27	0.9	16.3		
	V	762	5	543	2,715	73	73	0	73	219	0.7	2.5	3.83	3.67
New Mexico	VI	157	0	125	895	6	26	0	0	32	0.8	3.9	0.87	1.21
Missouri	VII	8,174	1,338	7,309	31,079	639	176	20	30	865	1.06	10.0	60.49	42.00
Colorado		992	37	880	8,722	70	37	5	0	112	0.92	8.2		
Utah		278	66	118	608	32	14	0	114	160	0.66	1.2		
Montana		35	0	24	120	7	4	0	0	11	0.69	2.2		
	VIII	1,305	103	1,022	9,450	109	55	5	114	283	0.86	4.0	7.87	12.77
California	IX	0	0	145	965	21	14	0	2	37	no mine	3.9	1.02	1.30
Idaho		1,677	289	1,400	6,614	151	125	1	0	277	1.0	6.1		
Washington		227	36	213	2,723	2	12	0	0	14	1.1	17.79		
	X	1,904	325	1,613	9,337	153	137	1	0	291	1.02	6.66	13.56	12.62
National total		18,122	1,866	12,428	73,989	1,234	926	26	3,679	5,865	0.79	2.44	100	100

Source: References 3 and 11.

TABLE 44

RATIO OF TOTAL WASTE ROCK-CONCENTRATOR DRY WASTES AND WASTEWATER TO ORE MINED FOR LEAD-ZINC AND ZINC ORES FOR SIC 1031 (METRIC)

State	EPA Region	Ore mined 10^3 TPY	Waste rock 10^3 TPY	Ratio of total waste rock to ore	Concentrator dry waste 10^3 TPY	Ratio of dry concentrator waste to ore	Concentrator wet waste 10^3 TPY	Ratio of wet waste to ore
New Jersey		183	0	0.00	123	0.67	615	3.36
New York		1,306	5	0.004	934	0.72	4,670	3.58
	II	1,489	5	0.003	1,057	0.71	5,285	3.55
Pennsylvania		544	0	0.00	406	0.75	2,265	4.16
Virginia		544	12	0.02	141	0.26	2,091	3.84
	III	1,088	12	0.01	547	0.50	4,356	4.00
Kentucky		59	0	0.00	0	0.00	-	NA
Tennessee		3,184	78	0.02	67	0.02	9,907	3.11
	IV	3,243	78	0.02	67	0.02	9,907	NA
Illinois		294	5	0.02	102	0.35	510	1.73
Wisconsin		468	0	0.00	441	0.94	2,205	4.71
	V	762	5	0.007	543	0.71	2,715	3.56
New Mexico	VI	157	0	0.00	125	0.80	895	5.70
Missouri	VII	8,174	1,334	0.16	7,309	0.89	31,079	3.80
Colorado		992	37	0.04	880	0.89	8,722	8.79
Utah		278	66	0.24	118	0.42	608	2.19
Montana		35	0	0.00	24	0.69	120	3.43
	VIII	1,305	103	0.08	1,022	0.78	9,450	7.24
California	IX	0	0	0.00	145	0.00	965	0.00
Idaho		1,677	289	0.17	1,400	0.83	6,614	3.94
Washington		227	36	0.13	213	0.94	2,723	12.00
	X	1,904	325	0.17	1,613	0.85	9,337	4.90
National total		18,122	1,866	0.10	12,450	0.69	73,989	4.08

Source: References 3 and 11.

The national ratio of waste rock to ore shows that 0.10 ton of waste rock was mined for every ton of ore. Dry concentrator waste totaled 0.69 ton for every ton of ore mined, while wet concentrator waste was 4.08 tons. The ratio of waste rock to ore varied from 0 to 0.24 ton/ton. One reason for this variation is the age of the mine. The states with the highest ratio of waste rock to ore are the states where new mines were recently opened. Very little waste rock is brought to the surface in the older mines because the rock is used underground for mine roads.

The ratio of dry concentrator wastes to ore mined for lead-zinc ranged from 0.26 ton/ton to 0.94 ton/ton. This variation in ratio is related to ore grade, percent of lead, and zinc and other metals present in the ore, rather than to differences in recovery methods and disposal of by-product materials by the mine. The variations in wet concentrator wastes reflect the difference in the mines, wet versus dry mines. Some mines must continually pump water out of the mine to be able to operate, while other mines are relatively dry. All of the water is pumped to the tailings pond with the concentrator wastes.

Mining Processes. Underground mining systems used in the New Lead Belt of southeast Missouri and in the Coeur d'Alene region of Idaho are used here as typical of lead-zinc mining practices.

In Missouri, lead-zinc ore occurs in the Bonne Terre dolomite as irregular masses of disseminated sulfides which vary greatly in area, shape, and thickness. Galena is the principal ore mineral, with lesser amounts of sphalerite and some copper sulfides. The ore-bearing formations are at depths of about 122 m (400 ft) in the northern part of the district, increasing in depth to about 396 m (1,300 ft) in the southern part. Ore bodies range in thickness from 12 to 30 m (40-100 ft) with the possibility of mineralizations on several horizons and sections of barren rock between masses of ore. This irregularity of ore bodies requires a flexible method of mining and the use of the random pillar arrangement to allow maximum ore recovery while leaving pillars of low grade ore and waste rock whenever possible.

Considerable groundwater is present throughout the area. Although the amount of influx varies, all mines must provide large sumps and pumping equipment with a capacity to handle several thousand gallons per minute.

Southeast Missouri New Lead Belt. All mines in the New Lead Belt district use the room and pillar method of mining with some differences in equipment and in methods of mining the ore face. There are about 10 operating mines in this area.[7]

Pillar size and spacing is determined by the distribution of ore. Pillars are generally 6 m (20 ft) wide, and headings have a maximum width of 10.5 m (34 ft). Some mines have mining levels about 2.3 m (8 ft) apart because of the manner of deposition of the ore, while the others are developed on single levels. Since the thickness of the ore varies greatly, several modifications of the mining system are employed. The maximum height of a single cut is about 7.0 m (23 ft), and the minimum working height is about 3.0 m (10 ft). For ore exceeding 6.1 m (20 ft) in thickness, the face may be advanced at the bottom and the high ore broken down by flat hole drilling from the top of the muck pile or by drilling from a truck-mounted aerial platform. Generally, the method preferred is to advance a face at the top of the ore and to mine the lower ore later in one or more benches. Extremely thick ore may be mined by advancing on two levels, then breaking out the ore separating these levels to form a single high room. In all cases, completely trackless mining is carried out, using diesel-powered equipment.

One mine in the southeast Missouri New Lead Belt differs from the others in that the haulage level is located below the ore body. Mining is carried out with diesel load-haul-dump equipment carrying broken ore a maximum distance of 274 m (900 ft) to raises which drop ore to the haulage level. Diesel-powered trains transport the ore to the shaft bottom. The sublevel is driven first, followed by a spiral incline at a 10 to 14 percent grade to the top of the ore body, where stoping commences. In this mine about 2,130 m (7,000 ft) of development openings were driven before production started, in contrast to the usual procedure in Missouri mines of starting production with a minimum of underground development.

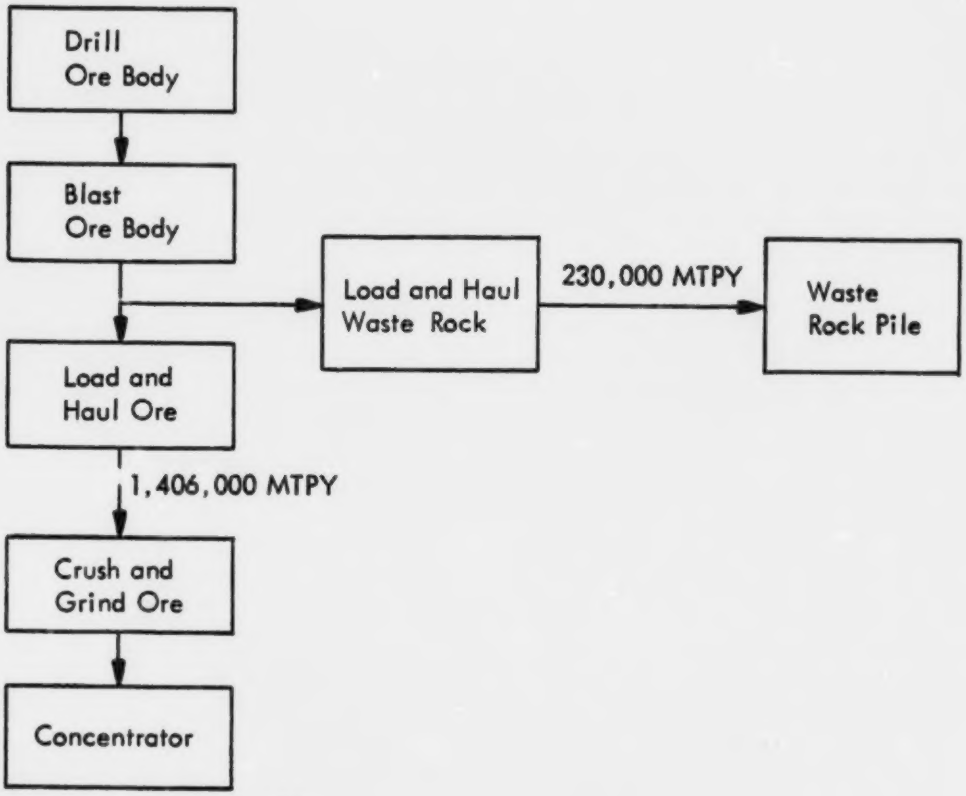
Figure 13 is a flow diagram of a typical lead-zinc mine in the southeast Missouri New Lead Belt.

All entries in the New Lead Belt are by vertical shafts. While almost all primary entries to the deposits are by conventional shaft sinking techniques, several bored shafts have been completed including ventilation shafts. A typical conventional shaft is 6 m (20 ft) in diameter and concrete lined. Some mines have smaller diameter entries.

In the New Lead Belt mines, some companies have continued to use conventional drum-type hoisting systems; others have installed friction-type hoists.

Drilling equipment may consist of rubber-tired, self-propelled diesel units carrying two or three drifters 10 to 11.5 cm (4-5 in.) in diameter capable of drilling faces up to 10.6 m (35 ft) wide by 9 m (30 ft) high. The newer methods favor the rotary percussion-type drill over the standard percussion type.

Dynamite is the standard explosive used for blasting in the district. Ammonium nitrate has not worked satisfactorily because the ground is too wet.


Drill Ore Body
Blast Ore Body
Load and Haul Waste Rock
230,000 MTPY
Waste Rock Pile
Load and Haul Ore
1,406,000 MTPY
Crush and Grind Ore
Concentrator

Source: Reference 11.

Figure 13. Typical underground lead-zinc mine (Missouri Lead Belt).

Mines in the New Lead Belt have committed themselves to the load-haul-dump concept. Principal units employed are the Joy transloader, the Wagner scoop-trans, Caterpillar front-end loaders, and Emico LHD units. All units are diesel powered and have rubber tires.

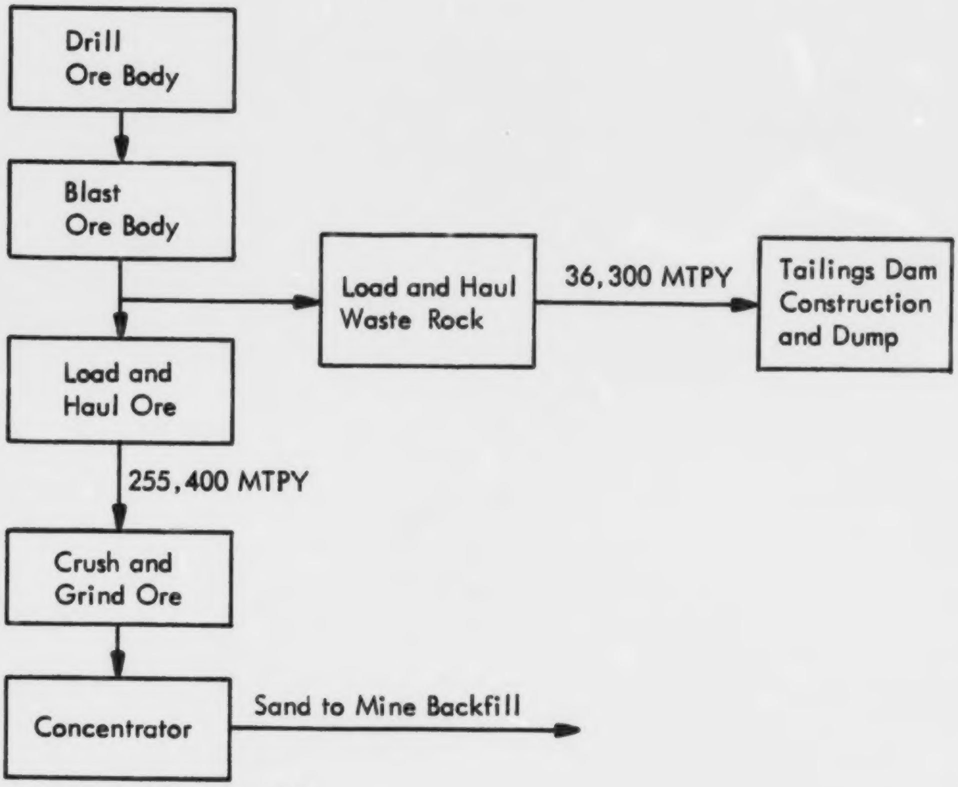
Bored vent shafts are used extensively for ventilation. Since the mines are not gaseous, the principal need for air is to clear the diesel fumes and the powder smoke. Air needs range from 2,800 to 11,320 m^3 (100,000-400,000 ft^3) per minute. Axivane fans are commonly used to circulate air through flexible ventilation tubing to the working faces in the mine.

Coeur d'Alene, Idaho, District. A flow diagram of mining for lead-zinc ore in the Coeur d'Alene region is shown in Figure 14. Lead-zinc producers in this region utilize mining practices which differ from those employed in Missouri. The entry to the mines is by train in a tunnel (adit) and then by hoist to the working levels. These companies have developed methods which incorporate the basic concept of cut-and-fill mining with a standard pillar pattern and sand backfilling of the stopes. The method has been termed the Bunker Hill pillar stoping system.

In this method, each 3-x 3-m (10-x 10-ft) pillar is carried vertically from footwall to hanging wall as mining progresses upward by cutting out one complete 2.43-m (8-ft) floor, filling with mill tailings, then cutting the next floor above. The only other support in addition to the pillar is an occasional timber set placed where needed. It is suspected that the pillars crush beneath the sandfill as successive floors are mined. The support that these pillars afford on the mining floor is sufficient to carry the load of the hanging wall.

Mining of a new floor begins with a raise-up between a chute and manway-service raise, as in horizontal cut-and-fill stoping. This is done to a full 2-m (7-ft) cut height with jackleg drills. Usually raise-ups of 9 to 18 m (30-60 ft) in length are sufficient to accomplish the objective of establishing the new floor. In more competent ground, main aisles of up to 45 m (150 ft) in length may be back-stoped in order to afford more working faces once the stope has been sandfilled and breasting begins on the new floor.

Stopes ideally are filled to within 0.3 m (1 ft) of the back of the old cut to allow breasting room. Poor ground conditions, however, often make it necessary to fill completely to the back of the cut. Care is taken to ensure that the stope is uniformly filled to the same elevation so that the back of the next cut will be flat, and the rubber-tired mucker will run on the same elevation. This requirement necessitates that sandfill outlets be strategically placed throughout the stope. In large pillar stopes, filling is conducted in two or three stages. As soon as a large area at one end of a stope cut is mined out, fill preparation is started.


Drill
Ore Body
Blast
Ore Body
Load and Haul
Waste Rock
36,300 MTPY
Tailings Dam
Construction
and Dump
Load and
Haul Ore
255,400 MTPY
Crush and
Grind Ore
Concentrator
Sand to Mine Backfill

Source: Reference 11.

Figure 14. Typical underground lead-zinc mine (Coeur d'Alene).

Laterals and aisles of the stope cut are driven in line. Rounds, either burn or breast down, depending on how tightly the stope has been filled, are drilled 3 m (10 ft) wide with jackleg machines. A standard drill pattern, electrical delay sequence, and explosive loading are used. This incorporates the use of smooth wall blasting techniques to minimize damage to the pillar and the arch.

A stope crew performs all drilling, blasting, face preparation, and muck removal. All fill preparation is done by a crew which is responsible for raising cribbed chutes and manways, building fill fences and traps, burlapping, and piping for sandfill.

Most raises and chutes are of the cribbed variety with a 1-x 1-m (3-x 3-ft) inside dimension. All chutes are generally inclined with the dip of the ore body or down to a 40° minimum.

Forced ventilation is required only in stopes which are not connected from level-to-level by a raise.

Mass Balances of Materials. Mass balance data for representative lead-zinc mining operations in Idaho and Missouri are shown in Figures 15 and 16.

Description of Individual Waste Streams. The only waste material involved in lead-zinc ore mining is waste rock.

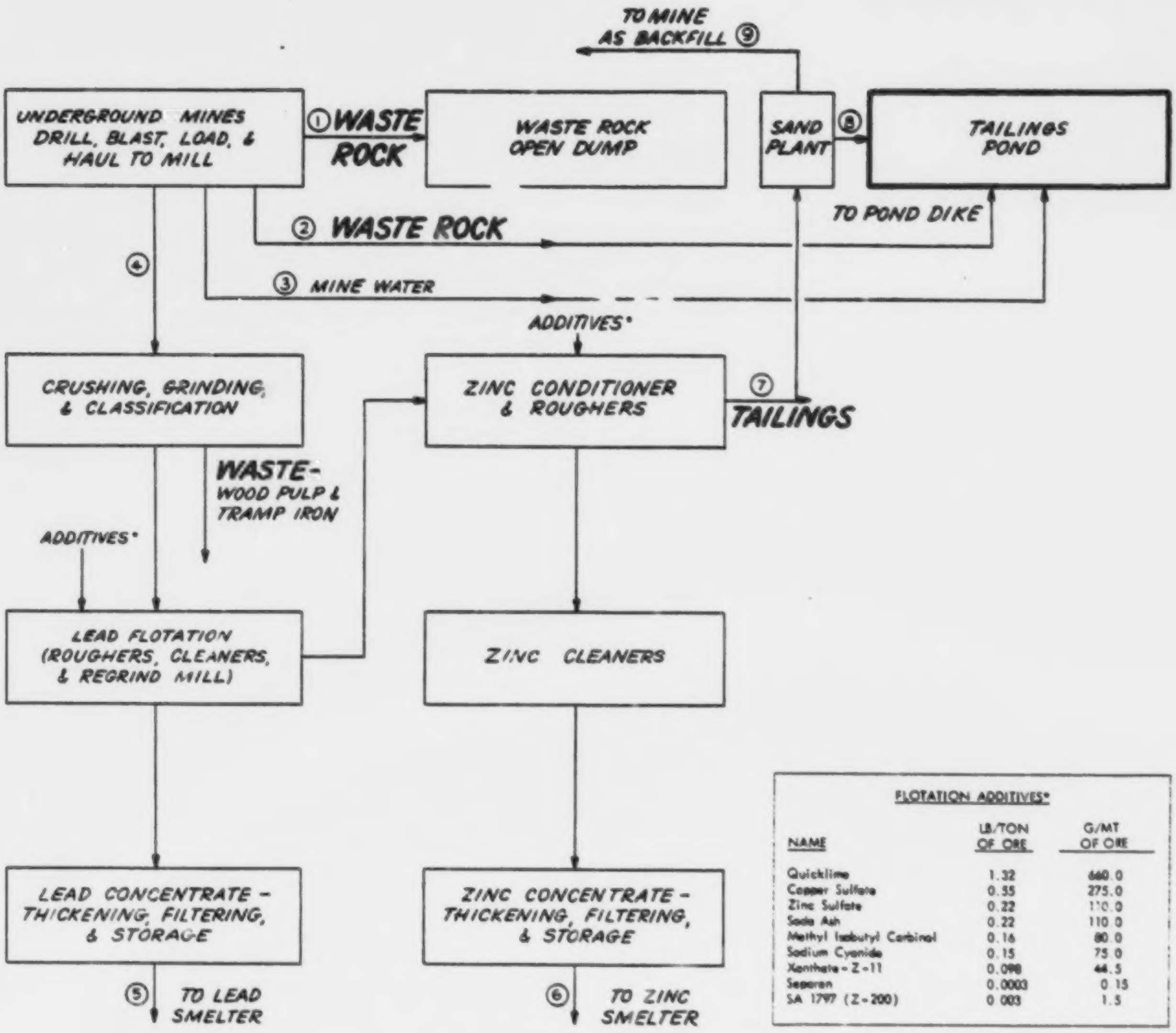
Identification of Potentially Hazardous Waste. Waste rock from the mining of lead-zinc ores is not considered to be potentially hazardous since the waste contains only trace amounts of lead and zinc not exceeding the background values for elements in the mining district.

Open Pit Mining Processes. There is no open pit mining of lead-zinc ores.

Concentrating Processes.

Flotation Processes. Two basic types of feed ores are processed by flotation to produce concentrate products: (1) lead-zinc sulfide ores; (2) lead-zinc-copper sulfide ores. The concentrating practices and wastes involved in processing these different feed ores are very similar. Flotation processes are used for all of them. Descriptions are given in the following sections for two typical examples of flotation processes, lead-zinc sulfide, and lead-zinc-copper sulfide ores.

Flotation of Lead-Zinc Sulfide Ores. A typical example of this type of operation is conducted at a flotation mill which produces lead and zinc concentrates containing silver. A simplified flow sheet for this flotation process is shown in Figure 15.


TO MINE
AS BACKFILL ⑨
UNDERGROUND MINES
DRILL, BLAST, LOAD, & HAUL TO MILL
① WASTE ROCK
WASTE ROCK OPEN DUMP
SAND PLANT
⑧
TAILINGS POND
TO POND DIKE
② WASTE ROCK
④
③ MINE WATER
ADDITIVES*
CRUSHING, GRINDING, & CLASSIFICATION
ZINC CONDITIONER & ROUGHERS
⑦
TAILINGS
WASTE-
WOOD PULP & TRAMP IRON
ADDITIVES*
LEAD FLOTATION (ROUGHERS, CLEANERS, & REGRIND MILL)
ZINC CLEANERS
LEAD CONCENTRATE - THICKENING, FILTERING, & STORAGE
ZINC CONCENTRATE - THICKENING, FILTERING, & STORAGE
⑤ TO LEAD SMELTER
⑥ TO ZINC SMELTER

FLOTATION ADDITIVES*

NAME	LB/TON OF ORE	G/MT OF ORE
Quicklime	1.32	660.0
Copper Sulfate	0.55	275.0
Zinc Sulfate	0.22	110.0
Soda Ash	0.22	110.0
Methyl Isobutyl Carbinol	0.16	80.0
Sodium Cyanide	0.15	75.0
Xanthate - Z-11	0.098	44.5
Separan	0.0003	0.15
SA 1797 (Z-200)	0.003	1.5

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS		① WASTE ROCK TO DUMP	② WASTE ROCK TO POND DIKE	③ MINE WATER	④ CRUDE ORE	⑤ LEAD CONC.	⑥ ZINC CONC.	(7a) TOTAL TAILINGS-DRY SOLIDS	(7b) TAILING LIQUID-TOTAL	⑧ TAILINGS SOLIDS TO POND	⑨ TAILING SOLIDS TO MINE BACKFILL
Quantity,	TPY	11.663	109.703		619,933	37.247	47.927	534.759		310.160	224.599
	Metric TPY	10.580	99.521		562.394	33,790	43,479	485.125		281.372	263.753
Quantity,	Million GPY			1548					612 - 792		
	Million Liters Yr			5859					2316 - 2998		
pH				3 - 5.7							
Assay, %	Pb				6.18	66.59	1.18	0.25			
	Zn				4.90	4.98	54.21	0.25			
	Ag, oz/T				3.94	40.45	2.83	0.18			
	Cu				0.06						
	As				0.07						
	Cd							0.0016			
	Sb				0.02						
	Fe				8.20						
	S				6.10						
	Mn				0.63						
	MgO				0.45						
	Al_2O_3				12.90						
	SiO_2				52.00						
	Other				4.55						

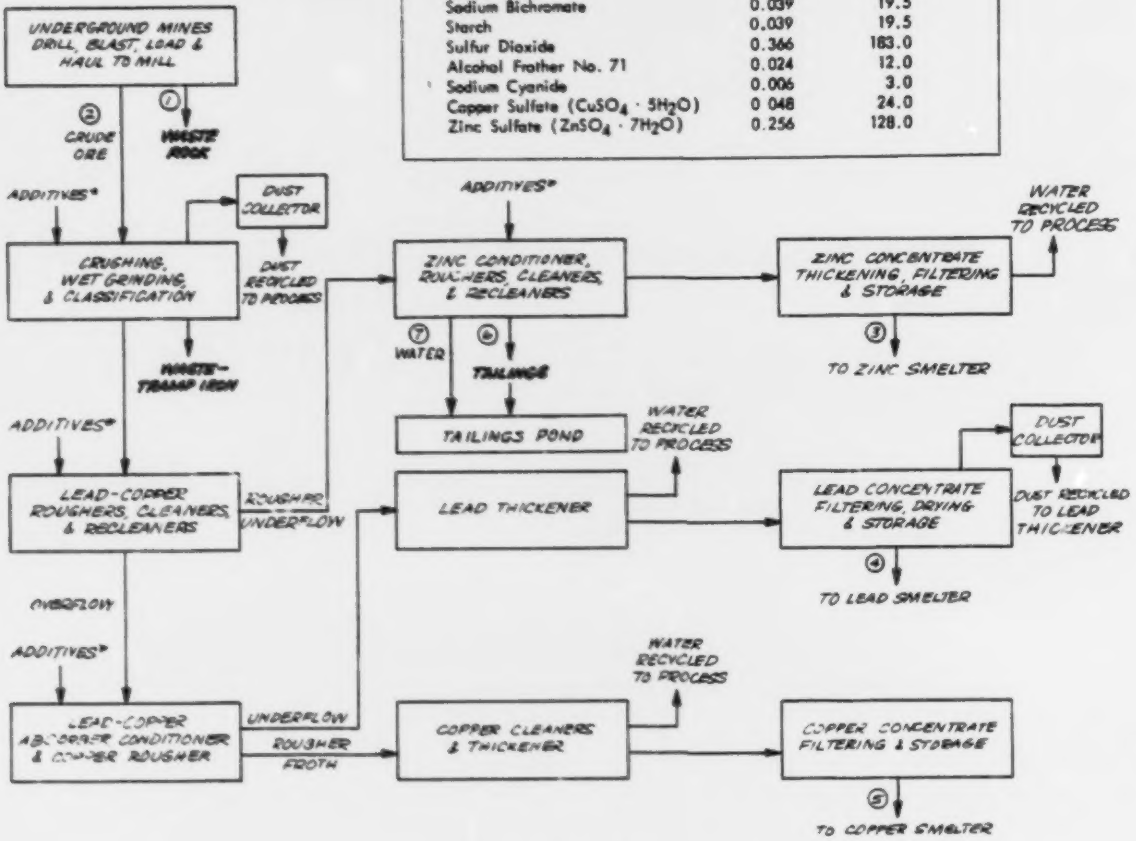
NOTE: Company operates lead and zinc smelters and refineries on site.

Source: Reference 11.

Figure 15. Mining and concentrating process: lead and zinc ores, Coeur d'Alene, Idaho district.

FLOTATION ADDITIVES*

NAME	LB/TON OF ORE	G/MT OF ORE
Sodium Isopropyl Zanthate No. 343	0.081	40.5
Z-200	0.026	13.0
Caustic Soda	0.013	6.5
Sodium Bichromate	0.039	19.5
Starch	0.039	19.5
Sulfur Dioxide	0.366	183.0
Alcohol Frother No. 71	0.024	12.0
Sodium Cyanide	0.006	3.0
Copper Sulfate ($CuSO_4 \cdot 5H_2O$)	0 048	24.0
Zinc Sulfate ($ZnSO_4 \cdot 7H_2O$)	0.256	128.0


UNDERGROUND MINES DRILL, BLAST, LOAD & HAUL TO MILL
①
WASTE ROCK
②
CRUDE ORE
ADDITIVES*
DUST COLLECTOR
CRUSHING, WET GRINDING, & CLASSIFICATION
DUST RECYCLED TO PROCESS
WASTE-TRAMP IRON
ADDITIVES*
LEAD-COPPER ROUGHERS, CLEANERS, & RECLEANERS
ROUGHER UNDERFLOW
OVERFLOW
ADDITIVES*
LEAD-COPPER ABSORBER CONDITIONER & COPPER ROUGHER
UNDERFLOW
ROUGHER FROTH
ADDITIVES*
ZINC CONDITIONER, ROUGHERS, CLEANERS, & RECLEANERS
⑦
WATER
⑥
TAILINGS
TAILINGS POND
WATER RECYCLED TO PROCESS
LEAD THICKENER
COPPER CLEANERS & THICKENER
WATER RECYCLED TO PROCESS
ZINC CONCENTRATE THICKENING, FILTERING & STORAGE
WATER RECYCLED TO PROCESS
③
TO ZINC SMELTER
DUST COLLECTOR
LEAD CONCENTRATE FILTERING, DRYING & STORAGE
DUST RECYCLED TO LEAD THICKENER
④
TO LEAD SMELTER
COPPER CONCENTRATE FILTERING & STORAGE
⑤
TO COPPER SMELTER

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS		① WASTE ROCK	② CRUDE ORE	③ ZINC CONC.	④ LEAD CONC.	⑤ COPPER CONC.	⑥ TAILINGS	⑦ WATER TO TAILINGS
Quantity,	TPY	10,660	1,065,828	52,800	64,800	6,952	941,276	2,800,000
	Metric TPY	9,671	966,903	47,899	58,786	6,307	853,911	2,560,000
Assay, %	Pb		4.42	2.33	72.7	4.98	0.093	
	Cu		0.20	0.40	0.89	27.60	0.018	
	Zn		2.60	52.2	1.91	0.68	0.083	
	Fe		1.59	3.04	3.38	27.10	1.30	
	Co		0.012	0.032	0.045	0.67	0.0045	
	Ni		0.017	0.031	0.070	1.06	0.0063	
	Ag, oz/T		0.33	16.0	1.86	1.20	0.025	
	CaO		19.0	2.03	0.54	0.33	20.1	
	MgO		10.3	1.03	0.17	0.13	11.4	
	Cd		0.012	0.79	0.035	0.014	0.0016	
	Mn		0.16	0.013	0.006	0.0035	0.18	
	Al_2O_3		0.52	0.17	0.12	0.037	0.56	
	S		1.86	28.92	16.07	33.25	0.46	
	Insol.		5.37	1.81	1.20	0.45	5.74	
	SiO_2		—	0.92	—	—	—	

Source: Reference 11.

Figure 16. Mining and concentrating process: lead-zinc-copper ore, southeast Missouri district.

Crushing, Grinding, and Classification. To reduce the ore size from about 30 cm (12 in.) to 1 cm (0.40 in.), a jaw crusher and cone crushers perform primary and secondary crushing. Tramp iron is removed by a separator magnet operating on or under a conveyor belt, and oversize from screening is recycled to the crushers.

Primary wet grinding is carried out in rod and ball mills operated in closed circuit with a spiral classifier. Classifier overflow is passed over a vibrating screen (4-mm (0.16-in.) openings) to remove wood pulp, which is discarded. The screen undersize is pumped to the lead flotation system.

Lead Flotation. The slurry conditioned by chemical additives (sodium cyanide, zinc sulfate, xanthate Z-11, methyl isobutyl carbinol, and soda ash) is processed in a flotation system consisting of rougher flotation cells and cleaner cells to produce coarse lead concentrate. Underflow from the roughers and cleaners is passed through cyclone classifiers to a ball-mill regrind circuit. The purpose of the regrind circuit is twofold: (1) to promote the recovery of the lead and zinc from the ore; (2) to reduce the amount of energy required to liberate the finely disseminated metal sulfides from the gangue material. From the regrind circuit, the slurry is returned as feed for the lead rougher flotation cells and is refloated.

After being dewatered by thickening and filtration, the final lead concentrate containing 66 to 70 percent Pb, 4 to 5 percent Zn, and about 1,370 g/MT of silver (40 oz/ton), is stored or shipped by railroad cars to a smelter. The classified fines overflow the hydroclone and are sent to the zinc flotation section of the plant.

Zinc Flotation. The fines slurry obtained by classification of the lead rougher underflow is sent to a zinc conditioner and mixed with reagent additives (xanthate Z-11, methyl isobutyl carbinol, copper sulfate, quicklime, and separan). From the conditioner, the slurry is sent to the zinc flotation machines consisting of rougher flotation cells and cleaner cells. The solids in the slurry average about 72 percent -200 mesh with 1 percent +48 mesh and about 57 percent -325 mesh. Tailings from the rougher cells are sent to a sand plant, and the coarse materials are used for mine backfill. The fines are disposed of on land in a tailings pond.

The rougher cell middlings are processed in a cyclone classifier; the coarse fraction (underflow) goes to the ball-mill regrind circuit; and the fine fraction is returned through the zinc conditioner to the rougher cells as feed material.

Froth from the zinc cleaner cells is dewatered by thickening and filtration to produce the final zinc concentrate. This concentrate, which contains 54.21 percent Zn, 1.18 percent Pb, and 88 g (2.83 oz) of silver per ton, is stored or shipped in railroad cars to a zinc smelter or electrolytic zinc refinery. The depressed material from the zinc cleaner cells is returned to the zinc rougher cells for more efficient product recovery.

Flotation of Lead-Zinc-Copper Sulfide Ores. A simplified flow sheet representative of this type of flotation process is shown in Figure 16.

Crushing, Grinding, and Classification. The crude ore is crushed by primary and secondary crushers to -2.54 cm (1 in.) size and then wet ground in rod mills to produce a 50 percent water slurry containing finely divided solids. A cyclone classifier separates the over-size which is returned to the rod mill. A dust collector is provided for the crushing operation, and the material collected in the cyclone is sent to the flotation circuit for metal recovery.

Flotation Operations. The slurry from the rod mill is sent to a flotation system (roughers, cleaners, and recleaners) where chemicals are added to produce a froth to activate the lead and copper minerals which are floated off as froth. The zinc materials which are not activated and the gangue from the ore flow to a separate zinc flotation circuit. The froth (containing lead-zinc minerals) from the lead-zinc flotation circuit is sent to a lead-copper conditioner where chemical reagents are added and the lead is depressed.

The conditioned froth passes to a copper rougher where the copper minerals are floated; the depressed lead passes as underflow to a separate lead processing system which includes thickening, filtering, and drying to yield a concentrate containing about 5 percent moisture. The dried lead concentrate, containing about 73 percent Pb (dry basis), is stored or shipped to a smelter.

The copper-containing froth from the copper roughing and cleaning operations is dewatered in a thickener and a filter. The final copper concentrate containing 10 percent moisture and about 28 percent Cu (dry basis) is stored or shipped to a smelter.

The copper-rougher underflow containing zinc minerals and gangue is conditioned and floated in roughers, cleaners, and recleaners to yield a crude zinc concentrate and tailings. The crude zinc concentrate is then dewatered in a thickener and filtered to yield a final zinc concentrate containing about 10 percent moisture and 52 percent Zn (dry basis). The final concentrate is stored or shipped to a smelter.

Mass Balance of Materials. Mass balance data for Example 1, lead-zinc sulfide ores, and Example 2, lead-zinc-copper sulfide ores, are shown on the table in Figures 15 and 16. For an ore consumption rate of 562,394 MT (619,933 tons) per year (MTPY), the quantities of waste material, tailings solids, are 485,125 MTPY (534,758 TPY).

Description of Individual Waste Streams. The only waste stream in the flotation operations is the tailings from the zinc rougher. In the example lead-zinc flotation plant, the concentrator tailings solids contain 0.25 percent lead, 0.25 percent zinc, and 0.0016 percent cadmium. In the case of the Pb, Zn, and Cu ores, the tailings could contain 0.2 percent copper. All these components are potentially hazardous substances and are commonly present in the tailings from lead-zinc flotation plants. Also, the feed ore may contain arsenic, a highly toxic material and, therefore, a potentially hazardous material. However, the arsenic normally floats with the copper and is recovered at the smelter refinery. No analytical data are available to establish whether any arsenic is present in the concentrator wastes. Table 45 is analytical data from the literature on concentrator tailings in the Coeur d'Alene. Table 46 is a compilation of all data reported by the companies contacted in the course of this program.

Identification of Potentially Hazardous Wastes. Tailings from all lead-zinc concentrator plants are classified as potentially hazardous wastes which require disposal safeguards to prevent environmental contamination. The concentrates from ores containing pyrite are generally considered to be more hazardous, so special care must be taken in their disposal.

Table 47 shows the total and potentially hazardous wastes from the mining and concentrating of lead zinc ores. In 1974, there were 11,955,000 MT dry weight (13,178,000 tons) and 61,817,000 MT wet weight (68,142,000 tons) of potentially hazardous wastes resulting from concentrating lead-zinc ores.

Tables 48 and 49, respectively, show the 1977 and 1983 projections for total and potentially hazardous wastes resulting from the mining and concentrating of lead-zinc ores.

Waste Treatment and Disposal

The waste treatment practices for wastes disposed on land by the lead-zinc industry are similar to the practices of the underground copper mining industry (Section I). The mine waste rock is either disposed of in waste rock piles, used for construction of tailings dams and mine roads, disposed of in the tailings ponds, or crushed for use as mine backfill. The wastes from concentrating operations are disposed of in tailings ponds, with some mines using the coarse fraction for mine backfill.

Mining Waste Treatment and Disposal. There is no generation of potentially hazardous waste resulting from the mining of lead-zinc ores in the United States.

TABLE 45

ANALYSIS OF TAILINGS FROM LEAD-ZINC MINES AND CONCENTRATORS (CONCENTRATION IN PPM)

Element	Concentrator 1	Concentrator 2	Concentrator 3	Concentrator 4	Background
Calcium	1,549	3,279	2,524	1,598	1,500
Cadmium	8.5	6.3	19.2	20.9	--
Copper	71.6	72.5	136	117.8	21
Iron	76,708	59,813	88,917	113,667	11,800
Potassium	335	406	422	287	1,300
Magnesium	2,460	2,482	3,781	2,680	3,700
Manganese	5,527	5,932	9,123	10,154	490
Sodium	92	69	75	69	151
Lead	2,302	2,624	4,380	4,462	51
Antimony	365	360	390	1,463	--
Zinc	2,057	1,316	3,547	3,366	150

Source: Reference 13.

TABLE 46

ANALYTICAL DATA FOR LEAD-ZINC TAILINGS SOLIDS FOR SIC 1031 (CONCENTRATION IN PPM)

Lead	Zinc	Copper	Cadmium
2,200	1,700	700	
7,800	1,500 600		
3,140	3,590		16
900	2,000 6,000		
400	2,000		
3,900	2,100	380	
2,000	1,300	150	

Source: References 7, 8, and 12.

TABLE 47

TOTAL AND POTENTIALLY HAZARDOUS WASTES FROM MINING AND CONCENTRATING OF LEAD-ZINC AND ZINC ORES FOR 1974 FOR SIC 1031 (METRIC)

		Dry waste weight (10^3 TPY)						Wet weight (10^3 TPY)	
			Total potentially hazardous waste	Waste rock		Concentrator tailings		Concentrator tailings	
State	Region	Total process waste		Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous
New Jersey		123	123	0	0	123	123	615	615
New York		939	934	5	0	934	934	4,670	4,670
	II	1,062	1,057	5	0	1,057	1,057	5,285	5,285
Pennsylvania		406	0	0	0	406	0	2,265	0
Virginia		153	141	12	0	141	141	2,091	2,091
	III	559	141	12	0	547	141	4,356	2,091
Kentucky		0	0	0	0	0	0	NA	NA
Tennessee		145	0	78	0	67	0	9,912	0
	IV	145	0	78	0	67	0	9,912	0
Illinois		107	102	5	0	102	102	510	510
Wisconsin		441	441	0	0	441	441	2,205	2,205
	V	548	543	5	0	543	543	2,715	2,715
New Mexico	VI	125	125	0	0	125	125	895	895
Missouri	VII	8,647	7,309	1,338	0	7,309	7,309	31,079	31,079
Colorado		917	880	37	0	880	880	8,722	8,722
Utah		184	118	66	0	118	118	608	608
Montana		24	24	0	0	24	24	120	120
	VIII	1,125	1,022	103	0	1,022	1,022	9,450	9,450
California	IX	145	145	0	0	145	145	965	965
Idaho		1,689	1,400	289	0	1,400	1,400	6,614	6,614
Washington		249	213	36	0	213	213	2,723	2,723
	X	1,938	1,613	325	0	1,613	1,613	9,337	9,337
National total		14,294	11,955	1,866	0	12,428	11,955	73,994	61,817

Source: References 3 and 12.

TABLE 48

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING OF LEAD-ZINC AND ZINC ORES FOR 1977 FOR SIC 1031 (METRIC)

		Dry waste weight (10^3 TPY)						Wet weight (10^3 TPY)	
				Waste rock		Concentrator tailings		Concentrator tailings	
State	EPA Region	Total process waste	Total potentially hazardous waste	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous
New Jersey		157	157	0	0	157	157	787	787
New York		1,202	1,196	6	0	1,196	1,196	5,978	5,978
	II	1,359	1,353	6	0	1,353	1,353	6,765	6,765
Pennsylvania		520	0	0	0	520	0	2,899	0
Virginia		195	180	15	0	180	180	2,676	2,676
	III	715	180	15	0	700	180	5,575	2,676
Kentucky		0	0	0	0	0	0	NA	NA
Tennessee		186	0	100	0	86	0	12,687	0
	IV	186	0	100	0	86	0	12,687	0
Illinois		137	131	6	0	131	131	653	653
Wisconsin		564	564	0	0	564	564	2,822	2,822
	V	701	695	6	0	695	695	3,475	3,475
New Mexico	VI	160	160	0	0	160	160	1,146	1,146
Missouri	VII	11,068	9,356	1,713	0	9,356	9,356	39,781	39,781
Colorado		1,173	1,126	47	0	1,126	1,126	11,164	11,164
Utah		235	151	84	0	151	151	778	778
Montana		31	31	0	0	31	31	154	154
	VIII	1,439	1,308	131	0	1,308	1,308	12,096	12,096
California	IX	186	186	0	0	186	186	1,235	1,235
Idaho		2,162	1,792	370	0	1,792	1,792	8,466	8,466
Washington		319	273	46	0	273	273	3,485	3,485
	X	2,481	2,065	416	0	2,065	2,065	11,951	11,951
National total		18,295	15,303	2,387	0	15,909	15,303	94,711	79,125

Source: References 3 and 12.

TABLE 49

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING LEAD-ZINC AND ZINC ORES FOR 1983 FOR SIC 1031 (METRIC)

		Dry waste weight (10^3 TPY)						Wet weight (10^3 TPY)	
				Waste rock		Concentrator tailings		Concentrator tailings	
State	EPA Region	Total process waste	Total potentially hazardous waste	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous
New Jersey		173	173	0	0	173	173	867	867
New York		1,324	1,317	7	0	1,317	1,317	6,585	6,585
	II	1,497	1,490	7	0	1,490	1,490	7,452	7,452
Pennsylvania		572	0	0	0	572	0	3,194	0
Virginia		216	199	17	0	199	199	2,948	2,948
	III	788	199	17	0	771	199	6,142	2,948
Kentucky		0	0	0	0	0	0	NA	NA
Tennessee		204	0	110	0	94	0	13,976	0
	IV	204	0	110	0	94	0	13,976	0
Illinois		151	144	7	0	144	144	719	719
Wisconsin		622	622	0	0	622	622	3,109	3,109
	V	773	766	7	0	766	766	3,828	3,828
New Mexico	VI	176	176	0	0	176	176	1,262	1,262
Missouri	VII	12,192	10,306	1,887	0	10,306	10,306	43,821	43,821
Colorado		1,293	1,241	52	0	1,241	1,241	12,298	12,298
Utah		25,	166	93	0	166	166	857	857
Montana		34	34	0	0	34	34	169	169
	VIII	1,586	1,441	145	0	1,441	1,441	13,524	13,524
California	IX	204	204	0	0	204	204	1,361	1,361
Idaho		2,381	1,974	407	0	1,974	1,974	9,326	9,326
Washington		351	300	51	0	300	300	3,839	3,839
	X	2,732	2,274	458	0	2,274	2,274	13,165	13,165
National total		20,155	16,857	2,631	0	17,523	16,857	104,332	87,162

Source: References 3 and 12.

Concentrator Waste Disposal Operation.

Flotation Tailings Disposal. Figures 17, 18, 19 are flow diagrams describing the tailings disposal methods at three typical lead-zinc concentrators. The first flotation plant (Figure 17) disposes of tailings in a tailings pond. Before the plant started up, an earthen dam was built to contain the tailings slurry. Then the land was cleared of vegetation and the roots grubbed out. The land was leveled by using a bulldozer and the soil compacted to present an impervious surface to the tailings slurry which is pumped to the pond.

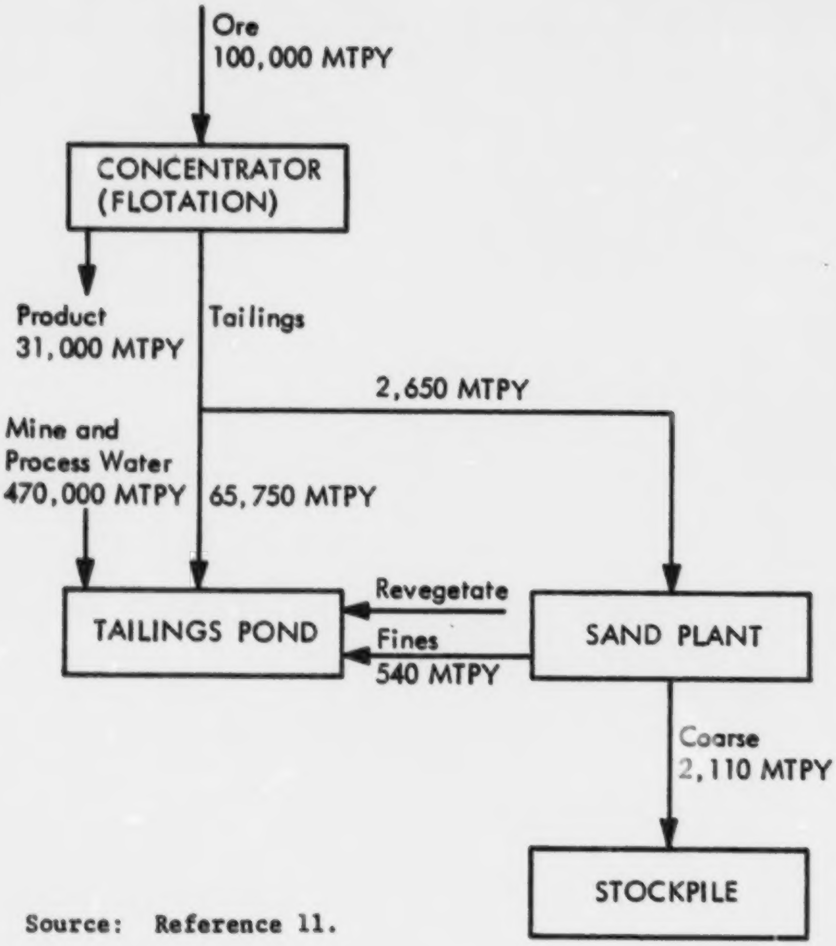
A portion (4 percent) of the tailings is sent to a sand plant where the coarse sand is collected and stockpiled. The fines are discharged to a settling pond, and the water discharged from the settling pond to a dispersion pond for final treatment before discharge. When a tailings pond is full, another tailings pond is put into the system, and the old pond is vegetated with native trees, shrubs, and grasses.

The second flow diagram (Figure 18) describes a system used in the Coeur d'Alene region. The flotation tailings are pumped to a sand plant where they are divided into two fractions. A coarse fraction (42 percent) is pumped back to the mine, mixed with cement, and used to backfill the mined-out stopes. The fine fraction (58 percent) is pumped to the tailings pond for disposal.

Waste rock is crushed and used for constructing the dam for the tailings pond. The tailings slurry is pumped to the pond and discharged about 3 m (10 ft) from the face of the dam. A water decant system was installed to recover the water from the pond so it could be pumped to a wastewater treatment plant. The effluent from the wastewater treatment plant is discharged, by state permit, into a river, and the sludge from the treatment plant is pumped to the tailings pond.

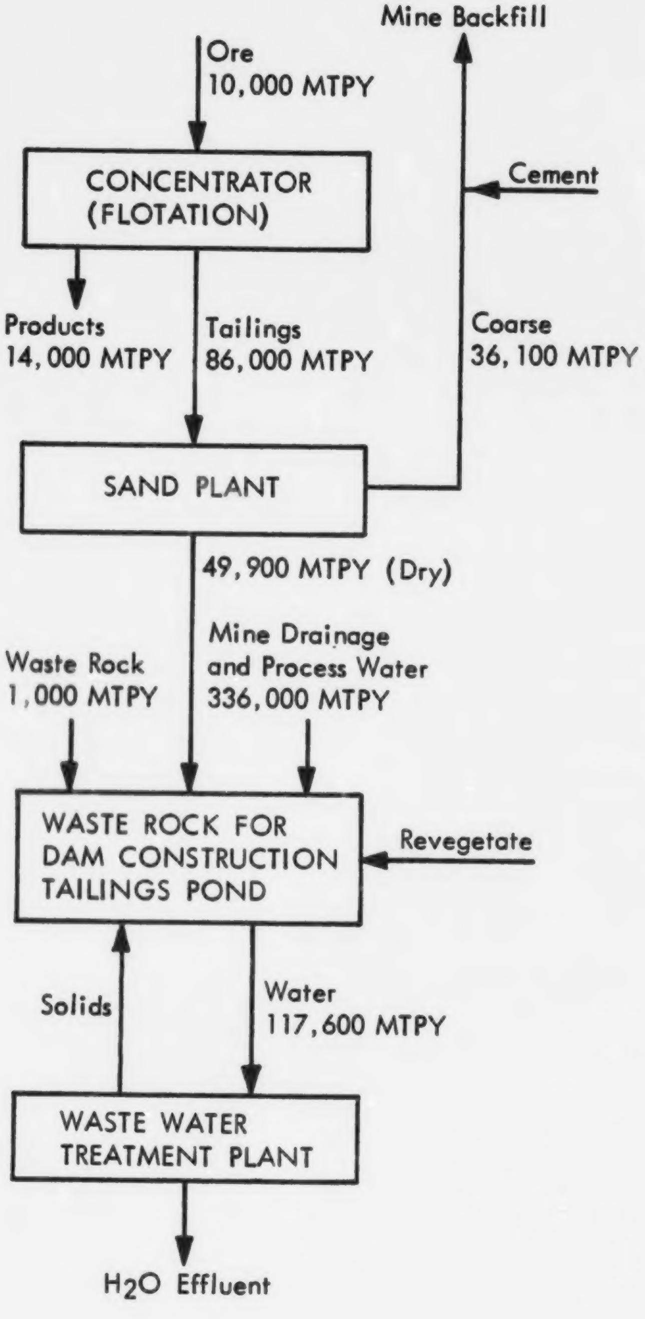
The acid mine water is also pumped to the tailings pond; the analysis of the solids in the mine drainage as well as the analysis of the tailings are shown in Tables 45 and 46. Pumping of acid mine water to the tailings pond is considered to be a poor practice; this is recognized by the company, and they are contemplating building a treatment plant to treat the mine water before it is pumped into the pond. Neutralization of the mine water before being added to the tailings pond will help to prevent hazardous materials from leaching out of the pond and appearing in the stream. The solubility of the metals is drastically reduced by raising the pH of the tailings slurry to above 8.

For stabilization, the dam at this plant is being revegetated with native shrubs. There are no plans for abandonment of this tailings pond during the life of the mine and concentrating plant. As more pond space is needed, the dam is raised with a waste rock and tailings mixture.


Ore
100,000 MTPY
CONCENTRATOR (FLOTATION)
Product
31,000 MTPY
Tailings
2,650 MTPY
Mine and Process Water
470,000 MTPY
65,750 MTPY
Revegetate
TAILINGS POND
Fines
540 MTPY
SAND PLANT
Coarse
2,110 MTPY
STOCKPILE

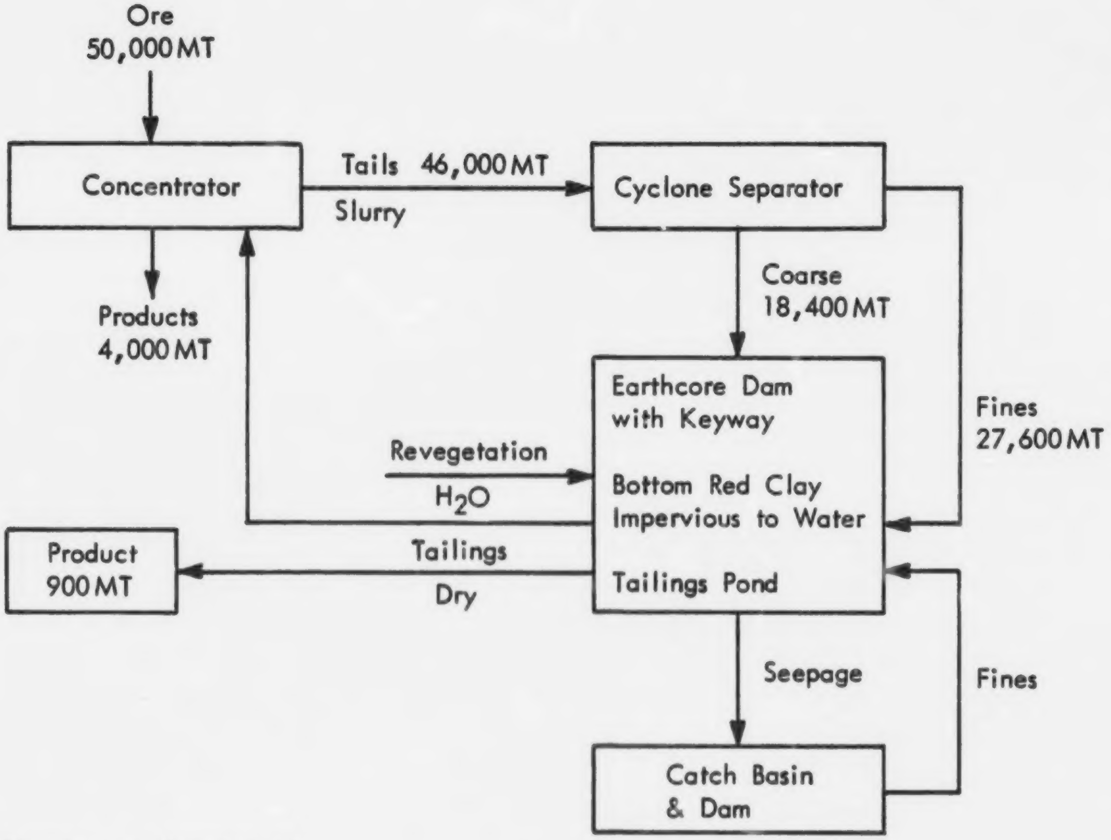
Source: Reference 11.

Figure 17. Level I technology for waste treatment and disposal, lead-zinc ores (Coeur d'Alene).


Ore
10,000 MTPY
Mine Backfill
CONCENTRATOR (FLOTATION)
Cement
Products
14,000 MTPY
Tailings
86,000 MTPY
Coarse
36,100 MTPY
SAND PLANT
49,900 MTPY (Dry)
Waste Rock
1,000 MTPY
Mine Drainage and Process Water
336,000 MTPY
WASTE ROCK FOR DAM CONSTRUCTION TAILINGS POND
Revegetate
Solids
Water
117,600 MTPY
WASTE WATER TREATMENT PLANT
H2O Effluent

Source: Reference 11.

Figure 18. Levels II and III technology for waste treatment and disposal, lead-zinc ores (Coeur d'Alene).


Ore
50,000 MT
Concentrator
Tails 46,000 MT
Slurry
Cyclone Separator
Coarse
18,400 MT
Products
4,000 MT
Earthcore Dam
with Keyway
Fines
27,600 MT
Revegetation
H2O
Bottom Red Clay
Impervious to Water
Product
900 MT
Tailings
Dry
Tailings Pond
Seepage
Fines
Catch Basin
& Dam

Source: Reference 11.

Figure 19. Level I technology for waste treatment and disposal, lead-zinc ores (southeast Missouri).

The third flow diagram, Figure 19, shows the procedure for disposal of flotation tailings in the southeast Missouri New Lead Belt.

The tailings are pumped to a cyclone separator. The coarse material (40 percent of the tailings) is discharged on the dam and used for dam construction. The fines (60 percent of the tailings) are discharged on the pond side (inside face) of the dam in the hope that they will seal the dam and prevent seepage.

The dams for these tailings ponds are earth-core dams, keywayed for strength and stability. The pond is in a valley, and the dam is built across the end of the valley and tied into the hills. The red clay valley floor in this area is impervious to water.

The construction of an earthen dam downstream from the tailings pond has formed a catch basin. Seepage water from the pond is caught in the basin and pumped back to the tailings pond.

A portion of the dry fines (2.5 percent) is sold to local farmers and used as agricultural lime.

The stream survey study conducted in this lead belt by the University of Missouri at Rolla concluded that there was no metal pollution from the tailings ponds.[8/] There was a problem with mine water, however, and this was solved. There is a problem with transportation of concentrates which has not yet been solved.

Levels I, II, and III Technology. Level I technology for disposal of flotation tailings in the lead-zinc industry is the use of tailings ponds for final disposal and the revegetation of these ponds. All of the lead-zinc mining and concentrating companies use tailings ponds and revegetate the dams for stabilization, Figures 17 and 19.

Level II technology for disposal of flotation tailings is the practice of backfilling the stopes in the mines with the coarse tailing in areas where this is practical. The tailings are pumped to a sand plant and separated into two fractions. The coarse fraction is pumped to the mine, mixed with cement, and used to fill mined-out stopes. The fines are pumped to a tailings pond. Approximately 25 percent of the companies are using Level II technology. An example of Level II technology is shown in Figure 18.

Level II technology for lead-zinc mines with acid mine water is neutralization of the acid mine water before discharge to the tailings pond. At present, one company out of the six which have acid water present is studying the use of this practice. Level II technology is equivalent to Level III technology, and an example is shown in Figure 18. About 17 percent of the companies are using Levels II and III technology.

The use of tailings ponds into which no unneutralized acid mine water is pumped, with revegetation of the pond and dam, is environmentally adequate. The only risk encountered is when unneutralized acid mine water (pH 2-2.5) is pumped to the pond, lowering the pH of the pond below 6. The solubility of some metal salts increases when the pH of the contacting solutions is below 7. Therefore, acid mine water must be treated to raise the pH above 8 before it is discharged to a tailings pond. The tailings slurry from the lead-zinc flotation is generally above 10 pH, and at this pH there is practically no solubility of the metal salts.9,10/

Lead, zinc, cadmium, copper, and bismuth are contained in the waste. These metals are potentially hazardous if the pH of the contacting solutions is below 8. The pH of the contacting solutions affects the potential hazard presented by the waste. There is no radioactive material in the wastes.

Based on the returned questionnaires, company data, and our engineering judgment, there were in 1974 approximately 11,900,000 MT dry weight (13,000,000 tons) of potentially hazardous waste resulting from the concentrating of lead-zinc ores (Table 47).

Future Adequacy of the Technology Identified. The adequacy of the Level II technology treatment methods will not be changed by air and water pollution enforcement regulations. There are no predicted changes in volume and composition of waste due to the future imposition of air and water pollution controls. There is one change that might occur if air pollution controls are imposed on the crushing and grinding plants. All plants which do not have dust collectors will probably change to wet grinding rather than install dust collectors. However, a waste material is not generated in this step. In fact, collectors and wet grinding are processing steps used to increase the feed to the concentrator.

The Level II technology identified can be retrofitted to all lead-zinc mines and concentrating facilities. However, the capital cost and energy requirement will be considerable where it is necessary to build a mine water treatment plant. The lead time would vary from 18 months to 2 years.

Energy and Cost Requirements. The energy amounts and cost associated with the proposed treatment and control technologies have been estimated as a portion of the total cost necessary to implement the recommended technologies, and included in the waste treatment and disposal costs discussion which follows next.

Waste Treatment and Disposal Costs - Lead-Zinc Mining

Disposal Cost--Technology Level I. Potentially hazardous waste disposal practices in lead-zinc mining (Technology Level I) utilize a tailings pond. A starter dam is built initially and a sand plant is used to raise the dam as the amount of tailings increases. The sides of the dam are vegetated to help reduce erosion. The pond used as the representative plant covers 40 hectares (100 acres). Land costs (assumed to be $12,350/hectare ($5,000/acre)) are a significant fraction of the total disposal costs. Therefore, three alternative cases are presented to show how different resale values and methods of accounting affect the disposal cost per metric ton.

Table 50 presents the major assumptions for technology Level I. Table 51 presents the result for the Coeur d'Alene area. In general, the disposal costs associated with lead-zinc are substantially higher than the same costs for uranium or copper mining. The costs range from $0.55 to $1.63/MT of potentially hazardous waste. Total industry costs, if all companies used the technology, would be 6.8 to 20.3 million dollars per year. The total cost would amount to 3.2 percent to 9.5 percent of the value added in mining and concentrating of lead-zinc ores.

Technology Level I differs between Coeur d'Alene and southeast Missouri. Mines at the Missouri location usually contain a catch basin or dam, as well as a tailings pond. The additional dam adds to the cost of construction. In addition, approximately four more hectares (10 acres) are needed for the catch basin. These items increase the cost of potentially hazardous waste disposal in Missouri.

Disposal Cost--Technology Levels II and III. In lead-zinc concentrating, waste disposal technology Level II is identical to technology Level III. Both levels utilize a wastewater treatment plant, as well as a tailings pond. Major cost assumptions of the treatment facility are presented in Table 52. Capital, operation, maintenance, and material costs are included. Table 53 presents the results for these more advanced technologies. Disposal of potentially hazardous waste material ranges in cost from $2.22 to $1.55/MT of waste. The cost to the industry would be 19.2 to 27.6 million dollars per year, or 8.9 percent to 12.9 percent of value added in mining.

TABLE 50

MAJOR COST ASSUMPTION - TAILINGS POND, LEAD-ZINC TECHNOLOGY LEVEL I (COEUR d'ALENE)

I. Capital

A. Land

Tailings pond assumed to have a maximum area of 40 ha (100 acres)*
Land cost at $12,350/ha ($5,000/acre) = $500,000
Levelized annual cost (assuming no resale value) = $58,500

B. Tailings Pump and Pipeline

Tailings pond assumed to be within 0.8 km (0.5 mile) of concentrator
Tailings pump and pipeline cost = $43,860†
Levelized annual cost at capital recovery factor of 0.117 = $5,100/year

C. Tailings Pond Starter Dam

Starter dam cost = $38,000†
Levelized annual cost = $4,446/year

D. Sand Plant

Sand plant capital cost = $58,000 †
Useful life = 10 years
Levelized annual cost = $9,406/year

II. Labor

Equivalent of one operator for sand plant and tailings pump and pipeline for 5 months; 8 hr/day at $8.97/hour‡, labor cost = $7,176/year

III. Supervision

Supervision cost (25% of labor cost)‡ = $1,794/year

IV. Maintenance

Assumed to equal 6%/year of nominal cost of equipment‡
Maintenance cost = $6,112/year

TABLE 50 (Concluded)

V. Insurance and Taxes

Insurance and tax (2% of nominal capital cost; land equipment)‡ = $12,797/year

VI. Energy and Power

Tailings pump and pipeline assumed to cause electricity with a value of less than $1,000/year§

VII. Vegetative Stabilization

Approximately 4 ha (10 acres) assumed to require stabilization§
Vegetative stabilization costs at $3,245/ha ($1,298/acre)** = $12,980
Levelized annual stabilization costs = $1,519/year

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[*] MRI estimate.

[†] Sears, M. B., et al. Correlation of radioactive waste treatment costs and the environmental impact of waste effluents in the nuclear fuel cycle for use in establishing as low as practicable guides - Milling of uranium ores. Oak Ridge National Laboratory, Oak Ridge, Tennessee, ORNL-TM-4903, v.1, May 1975. p. 187-188. Obtained by ratio of material handled (dry millings and water) to total cost for pump and pipeline.

[‡] See corresponding table for copper mining.

[§] MRI estimate.

[**] Ludeke, K. Vegetative stabilization of tailings disposal berms. in: Mining Congress Journal, January 1973. p. 39.

TABLE 51

DISPOSAL COSTS, POTENTIALLY HAZARDOUS WASTE FROM LEAD-ZINC TAILINGS POND, TECHNOLOGY LEVEL I - COEUR d'ALENE
(IN EQUIVALENT ANNUAL 1973 DOLLARS)

	Case "A"*	Case "B"†	Case "D"‡
Capital			
Land	$58,500	$48,906	0
Equipment and dam	18,867	18,867	$18,867
Labor	7,176	7,176	7,176
Supervision	1,794	1,794	1,794
Maintenance	6,122	6,122	6,122
Insurance and taxes	12,797	12,797	377
Energy and power	1,000	1,000	1,000
Vegetative stabilization	1,519	1,519	1,519
Total	$107,765	$98,171	$36,555
Metric ton of potentially hazardous waste deposited in tailings pond/year	$66,290	$66,290	$66,290
Cost/metric ton of potentially hazardous waste	$1.63	$1.48	$0.55
Total cost if entire industry adopted ($$10^6$/year)	$20.3	--	$6.8
Total cost as a percent of value added in mining	9.5%	--	3.2%

* Case "A" assumes all land required is purchased in year 1 and has no resale value after 20 years.

† Case "B" assumes all land required is purchased in year 1 and resold at same price after 20 years.

‡ Case "D" assumes land used for tailings pond has no opportunity cost to the company.

TABLE 52

ADDITIONAL COST ASSUMPTIONS - LEAD-ZINC MINING TECHNOLOGY LEVELS II AND III

A. Capital

The plant will handle 117,000 MT of water per year or 321,237 liters/day (84,862 gal/day).

The plant contains two systems; agitated open tank and sedimentation system.

1. Agitated tank capital cost 6,700 liter capacity (1,800 gal) = $40,000.* Useful life = 20 years.*
2. Sedimentation system capital cost = $11,000† (84,862 gal/day). Useful life = 40 years.
   Useful SL depreciation, net capital cost = $10,100.
3. Conveyor transport of solids = $15,000.‡
4. Equivalent annual cost (tank, conveyor and sedimentation) = $7,617/year.

B. Operation and Maintenance (includes energy and power)

1. Agitated tank, labor cost = $20,000/year.§ Agitated tank, maintenance cost $4,000/year.
2. Sedimentation system, labor cost = minimal.**
   Sedimentation system, maintenance cost = $1,155/year. **

C. Materials

The kilograms of dolomitic lime required per day.†
Lime cost (at $22/MT)†† approximately equal $100/year.

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* Blecker, H., and T. Nichols. Capital and operating costs of pollution control equipment modules. Vol. II--Data Manual. Environmental Protection Agency EPA-R5-73-0236. July 1973. p. 10.

† Blecker, H., and T. Nichols. Capital and operating costs of pollution control equipment modules. Vol. II--Data Manual. Environmental Protection Agency EPA-R5-73-0236. July 1973. p. 126-127.

‡ MRI estimate.

§ Blecker, H., and T. Nichols. Capital and operating costs of pollution control equipment,modules. Vol. II--Data Manual. Environmental Protection Agency EPA-R5-73-0236. July 1973. p. 11.

** Blecker, H., and T. Nichols. Capital and operating costs of pollution control equipment modules. Vol. II--Data Manual. Environmental Protection Agency EPA-R5-73-0236. July 1973. p. 127.

†† U.S. Environmental Protection Agency. Development document for effluent limitation guidelines and new source performance standards for the major inorganic produces. EPA-440/1-74-007-a. March 1974. p. 275.

TABLE 53

DISPOSAL COSTS, POTENTIALLY HAZARDOUS WASTE - LEAD-ZINC MINING, TAILINGS POND AND WASTEWATER TREATMENT PLANT, TECHNOLOGY LEVELS II AND III

	Case "A"*	Case "B"†	Case "D"‡
Capital			
Wastewater treatment plant	$ 7,617	$ 7,617	$ 7,617
Land	58,500	48,906	0
Other equipment and dam	18,867	18,867	18,867
Labor	27,176	27,176	27,176
Supervision	6,794	6,794	6,794
Maintenance	11,267	11,267	11,267
Materials	100	100	100
Insurance and taxes	14,117	14,117	14,117
Energy and power§	1,000	1,000	1,000
Vegetative stabilization	1,519	1,519	1,519
Total	$146,957	$137,363	$102,674
Metric tons of potentially hazardous waste	66,290	66,290	66,290
Cost/metric ton of potentially hazardous waste	$2.22	$2.07	$1.55
Total cost if entire industry adopted ($$10^6$/year)	$27.6	--	$19.2
Total cost as a percent of value added in mining	12.9%	--	8.9%

---

* Case "A" assumes all land purchased in year 1 with no resale value after 20 years.

† Case "B" assumes all land purchased in year 1 and resold at the same value after 20 years.

‡ Case "D" assumes land used for tailings pond has no opportunity cost to the company.

§ Not including energy and power requirements of wastewater treatment plant.

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SECTION III

ZINC ORES (SIC 1031)

Industry Characterization

Zinc Mining. The metallurgy of zinc began as an alloy material long before it became known as a metal. The oldest known piece of zinc is in the form of an idol found in the prehistoric Dacian settlement at Doroseh, Transylvania. The Romans were well acquainted with brass, an alloy of zinc, as early as 200 B.C. Zinc appears to have been known in India as early as 1000 to 1300 A.D. and was probably smelted commercially in the 14th Century.

About 1730, the technology of smelting zinc was brought from China to England; in 1739, a patent was obtained for a distillation method, and a smelter was erected at Bristol, England, with a mentioned capacity of 180 MT (198 tons) of zinc annually.

In the United States the first zinc was produced in 1835 at the Arsenal in Washington, D.C. The U.S. Government imported workers from Belgium and built a small zinc furnace utilizing zincite ore from New Jersey. The primary purpose of the furnace was to provide zinc to form brass for manufacture of standard units of weight and measure.

The U.S. zinc industry began in 1860 upon successful operation of a plant at LaSalle, Illinois, for the treatment of Wisconsin ores and also a plant at South Bethlehem, Pennsylvania, the same year. The processing technology of zinc improved rapidly with adoption of horizontal retorts, mechanically rabbled roasters, successful hydrometallurgy techniques to produce, by leaching, a zinc sulfate for lithopone, and the American process for production of zinc oxide directly from ore. In 1895, natural gas was discovered in Kansas, which led to the building of a number of smelters in the southwestern United States utilizing natural gas as a fuel.

The commercial introduction of the froth flotation process early in the 20th Century was the most significant technological advance in the zinc industry. The problems of smelting lead-zinc sulfide ores and concentrates had previously prevented efficient recovery of the zinc at lead smelters because the zinc ended up in the slag. The flotation process also made it possible to recover the zinc in mixed copper-lead-zinc ores.

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The zinc industry is an international basic industry with world-wide influence in mining, smelting, and trade. Canada is the world's largest producer with more than double the output of the Soviet Union, followed by the United States. Other large zinc-producing nations include Australia, Peru, Mexico, and Japan.

Domestic Production and Capacities. Domestic production of recoverable zinc was 433,000 MT (477,000 tons) in 1973 compared to 480,000 MT (529,000 tons)* in 1968 (Table 54). While Government data are not available for 1974, it is estimated that zinc ore concentrate production amounted to 1,138,000 MT (1,254,000 tons) (Table 55). Zinc production in 1974 was reported in 16 states. New Mexico led the nation with an estimated 235,500 MT (260,000 tons) of recovered zinc concentrate† product, followed by Missouri, Tennessee, and New York. The total zinc ore production by year is given in Table 56. The mine production of recoverable zinc in the United States, by state and year, is given in Table 57. Table 58 shows the mine production of zinc by EPA regions for 1973.

TABLE 54

U.S. ZINC PRODUCTION, 1968-1983
(RECOVERABLE CONTENT OF ORE)*

Year	Metric tons
1968	480,313
1969	501,794
1970	484,568
1971	455,907
1972	433,930
1973	432,734
1977	487,000†
1983	581,000†

Source: Reference 1.
* Data reported in tons and calculated to metric tons.
† Estimated production.

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* Recoverable zinc = the amount of pure zinc recovered.

† Recovered zinc concentrate = the amount of concentrate produced before smelting and refining of the ore.

TABLE 55

U.S. ZINC CONCENTRATE PRODUCTION - 1974*
(RECOVERED ZINC CONCENTRATE PRODUCTS)†

State	Metric tons
California	13,753
Colorado	37,105
Idaho	125,367
Illinois	60,963
Kentucky	2,268
Missouri	176,056
Montana	4,442
New Mexico	235,484
New York	59,439
New Jersey	146,446
Pennsylvania	54,431
Tennessee	156,354
Utah	13,608
Virginia	28,123
Washington	11,793
Wisconsin	12,156
U.S. total	1,137,788

EPA regions	Metric tons
Region I	--
Region II	205,885
Region III	82,554
Region IV	158,622
Region V	73,119
Region VI	235,484
Region VII	176,056
Region VIII	55,155
Region IX	13,753
Region X	137,160

Source: Reference 2.

* Engineering & Mining Journal and unpublished mining company data.

† Data reported in tons and calculated to metric tons.

TABLE 56

TOTAL ZINC MINE PRODUCTION OF ORE BY YEAR (ORE MINED)*

Year	1,000 Metric tons
1969	501,785
1970	484,560
1971	455,899
1972	433,922
1973	434,405

Source: Reference 3.

* Data reported in tons and calculated to metric tons.

TABLE 57

MINE PRODUCTION OF RECOVERABLE ZINC IN THE UNITED STATES BY STATE
(METRIC TONS)*

State	1969	1970	1971	1972	1973
Arizona	8,200	8,725	7,040	9,172	7,644
California	3,018	3,187	2,724	1,090	18
Colorado	48,729	51,431	55,502	57,879	52,924
Idaho	50,711	37,241	40,894	35,059	41,827
Illinois	12,487	15,237	11,526	10,321	4,762
Kansas	1,723	1,075	--	--	--
Kentucky	4,525	3,800	4,779	1,614	247
Maine	6,929	8,268	5,307	5,279	17,817
Missouri	37,284	46,013	43,739	56,175	74,706
Montana	5,572	1,321	327	10	66
Nevada	853	115	64	--	--
New Jersey	22,748	26,020	27,194	34,560	29,961
New Mexico	22,051	15,060	12,663	11,553	11,182
New York	53,277	53,140	57,533	55,110	73,894
Oklahoma	2,489	2,404	--	--	--
Pennsylvania	29,968	26,810	24,891	16,641	17,106
South Dakota	--	1	--	--	--
Tennessee	112,973	107,283	108,222	92,280	58,215
Utah	31,662	31,468	23,315	19,824	15,240
Virginia	16,967	16,386	15,267	15,230	15,134
Washington	8,834	10,846	5,245	5,881	5,786
Wisconsin	20,775	18,718	9,656	6,235	7,867
Other States	--	--	2	--	--
Total	501,785	484,560	455,899	433,922	434,405

Source: Reference 3.

* Data reported in tons and calculated to metric tons.

TABLE 58

MINE PRODUCTION OF RECOVERABLE ZINC IN THE UNITED STATES BY EPA REGION - 1973

Region	Metric tons*
I	17,817
II	103,857
III	32,241
IV	58,463
V	12,629
VI	11,183
VII	74,707
VIII	68,231
IX	7,663
X	47,614

Source: Reference 3.

* Data reported in tons and calculated to metric tons.

In 1972, the 25 leading zinc mines accounted for 89 percent of the domestic mine production. The five leading mines produced 36 percent; the first 10 together, 56 percent.

The U.S. mine capacity is not sufficient to meet the total requirements of the nation, and the U.S. dependence on imports for primary supply of zinc approximates 50 percent. Ores and concentrates are imported from throughout the world, with Canada supplying 60 percent, Mexico 24 percent, Peru 8 percent, and other countries, about 8 percent of the imports. The demand for zinc is expected to increase at 3 percent annually through 1983. The projected rate of demand would increase U.S. zinc production to 487,000 MT (537,000 tons) of metal in 1977 and 581,000 MT (640,000 tons) in 1983.

Number, Location, Size, and Age of Mines and Concentrators. The domestic zinc mining industry in 1974 comprised eight mining operations with a total of eight mines and seven concentrators (Table 59).[1/] Zinc mines vary in size from medium with less than 100 employees to large with over 500 employees. Recent data from the Engineering & Mining Journal and the Mining Enforcement and Safety Administration indicate that 88 percent of the operations employed more than 100 workers in 1974.

TABLE 59

NUMBER, LOCATION, AND SIZE OF ACTIVE ZINC MINING AND CONCENTRATING OPERATIONS

	Employees				Total	Total No.	Total No.
State	50-99	100-249	250-499	500-999	operations	mines	concentrators
Pennsylvania		1			1	1	1
Tennessee	1	4		1	6	6	5
New Jersey		1			1	1	1

Source: References 2 and 5.

The active zinc mines have been in operation for many years, some more than 90 years. The owners and operators have changed, but the mines have stayed in production. In Tennessee, two companies have recently purchased operating mines and concentrators from the former operators.

Lead and zinc statistics are combined because it is difficult to separate lead and zinc mining operations, as they are often mined together. However, approximately 50 percent of the domestic zinc production is from ores designated as zinc ores. These are ore deposits found principally in New Jersey, Tennessee, and Pennsylvania (Figure 12). The domestic mines operated essentially at capacity during 1972, but smelter production was down (Tables 60 and 61).

Employment. Total estimated employment in the eight active zinc mines in 1974 was approximately 1,800 according to figures issued by Engineering & Mining Journal, Mining Enforcement and Safety Administration, and from on-site investigations made by MRI. More than 65 percent of the zinc workers were employed in Tennessee (Table 59). Employment by region is shown in Table 62.

By-Products and Coproducts. Zinc production affects, and in turn is affected by, the demand for a variety of coproducts and by-products. Ores containing zinc also contain varying amounts of other valuable and recoverable materials including cadmium, copper, fluorspar, gallium, germanium, gold, indium, lead, manganese, silver, sulfur, and thallium. Zinc ranges from the major product, as in the Tennessee, Pennsylvania, and New Jersey deposits, to a coproduct as in the complex western ores and in the Missouri Lead Belt.

TABLE 60

U.S. ZINC CAPACITY AND PRODUCTION, 1972
(1,000 MT)

	Capacity	Production (recoverable content of ore)
Mine	494	478
Smelter	603	541

Source: Reference 7.

TABLE 61

U.S. ZINC PRODUCTION CAPACITY, 1972-1974
(1,000 MT)

	Production (recoverable content of ore)		
	1972	1973	1974
Mine	478	479	492
Smelter	633	541	540

Source: Reference 7.

TABLE 62

EMPLOYMENT AT ACTIVE ZINC MINE AND CONCENTRATOR OPERATIONS IN 1974

State	Employment
Pennsylvania	207
Tennessee	1,328
New Jersey	207
EPA regions	**Employment**
III	207
IV	1,368
II	207

Source: References 2 and 5.

The major products associated with zinc and recovered at zinc plants in stack gases, flue dusts, and residues are sulfur, cadmium, germanium, thallium, indium, and gallium. Zinc by-product and coproduct relationships are identified in Table 63.

Waste Generation and Characterization

Data on the production statistics by state and region for zinc ores are shown in Table 64. Data on the ratio of waste rock, overburden, and concentrator wastes to ore mined by state and region for SIC No. 1031 (zinc ores) are shown in Table 65.

The total national production of zinc ore in 1974 amounted to 3,912,000 MT (4,312,241 tons). The total national concentrator products for 1974 amounted to 3,272,000 MT (3,607,000 tons). There were 270,224 MT (297,871 tons) of zinc concentrate and 3,002,000 MT (3,309,000 tons) of miscellaneous products. The corresponding generation of waste rock was 79,000 MT/year (87,000 tons/year), overburden (0), and concentrator waste (tailings 596,000 MT/year (657,000 tons/year). The national ratio of waste rock to ore was only 0.02, and the ratio of concentrator waste to ore was 0.15.

TABLE 63

ZINC BY-PRODUCT AND COPRODUCT RELATIONSHIPS

Ore source	By-product and coproduct	Quantity* (MT)	Total product output (%)
Zinc	Cadmium	1,718	100.0
Zinc	Germanium	21,800	100.0
Zinc	Indium	6.9	100.0
Zinc	Thallium	1,181	100.0
Zinc	Gallium	W	W
Zinc	Manganese	5,000	16.7
Zinc	Silver	124.1	12.6
Zinc	Sulfur	267,000	3.0
Zinc	Gold	0.6	1.4
Zinc	Calcium	679,000	0.8
Zinc	Copper	4,500	0.4
Zinc	Stone	W	W
Zinc	Iron	4,500	--
Lead	Zinc	51,000	10.7
Fluorine	Zinc	10,800	2.2
Copper	Zinc	8,100	1.8
Silver	Zinc	1,800	0.4
Gold	Zinc	4.5	--

Source: Reference 4.

W = Withheld to avoid disclosing individual company confidential data.

* Data calculated to metric units.

TABLE 64

PRODUCTION STATISTICS BY STATE AND EPA REGION FOR ZINC ORES IN 1974 FOR SIC 1031 (METRIC)

State	EPA Region	Ore mined (10^3 TPY)	Wastes (10^3 TPY)				Products (TPY)			Ratio of		Total waste (%)	Concentrator wet waste (10^3 TPY)
			Waste rock	Over-burden	Concentrator wastes	Total waste	Zn conc.	Miscellaneous products	Total products	Total waste to ore	Total waste to product		
New Jersey	II	183	0	0	125	123	59,439	0	59,439	0.68	2.1	18.28	736
Tennessee	IV	3,184	79	0	67	146	156,354	2,961,126	3,117,480	0.05	0.05	21.64	7,915
Pennsylvania	III	545	0	0	406	406	54,431	40,823	95,254	0.75	4.3	60.08	2,094
National total		3,912	79	0	596	675	270,224	3,001,949	3,272,173	0.17	0.205	100.00	10,642

Source: References 2, 6, and 7.

TABLE 65

RATIO OF TOTAL WASTE ROCK--OVERBURDEN--CONCENTRATOR WASTES TO ORE MINED FOR SIC 1031, ZINC ORE (METRIC)*

State	EPA Region	Ore mined (10^3 TPY)	Waste rock (10^3 TPY)	Ratio of total waste rock to ore	Concentrator wet waste (10^3 TPY)	Ratio of wet waste to ore	Concentrator dry waste (10^3 TPY)	Ratio of concentrator dry waste to ore
New Jersey	III	183	0	-	736	4.02	125	0.673
Pennsylvania	III	545	0	-	2,094	3.84	406	0.745
Tennessee	IV	3,184	79	0.025	7,915	2.49	67	0.021
National		3,912	79	0.020	10,642	2.72	596	0.152

* Compiled from data in Table 64.

On a regional basis, the ratio of waste rock plus overburden to ore ranged from 0.020 to 0.025. The ratio of concentrator waste to ore ranged from 0.02 in Region IV to 0.75 in Region III. No overburden was produced in any of the operating regions, and Regions II and III also had no waste rock.

The zinc mines are underground units and are all located in the eastern half of the nation (Tennessee, Pennsylvania, and New Jersey). A discussion of typical mining and concentrating operations for zinc ores is presented in the following subsections. These operations are generally similar to those used for lead-zinc ores.

Mining Processes.

Underground Mining Process. In one typical zinc ore deposit, the zinc exists principally as zinc sulfide (sphalerite). Other minerals found in the ore deposit include zinc carbonate, dolomite, and limestone. Cadmium is also present in this ore.

As shown in Figure 20, the basic mining method consists of drilling, blasting with dynamite or AN-FO,* loading the broken ore by mobile load-haul units, hoisting ore and waste rock to the surface by skip hoist, and transporting ore by dump truck to a crushing plant.

Waste rock removed from the mine is deposited on an open stockpile. In this example, the rock is periodically removed from the stockpile, crushed and ground, and sold as a ground rock product for road-building uses. Following the initial startup period for these mines, there is no overburden waste.

Mass Balance of Materials. Figure 21 contains mass balance data for a zinc mine and concentrator. It shows that 454,000 MT (500,000 tons) of ore were mined for 27,216 MT (30,000 tons) of zinc concentrate recovered.

Description of Individual Waste Streams. The only waste stream in domestic mining of zinc ores is waste rock, and it is not considered to be potentially hazardous.

Concentrator Processes.

Flotation Process. All domestic zinc ores are concentrated by flotation. A representative flotation operation including available production data is shown in Figure 21.

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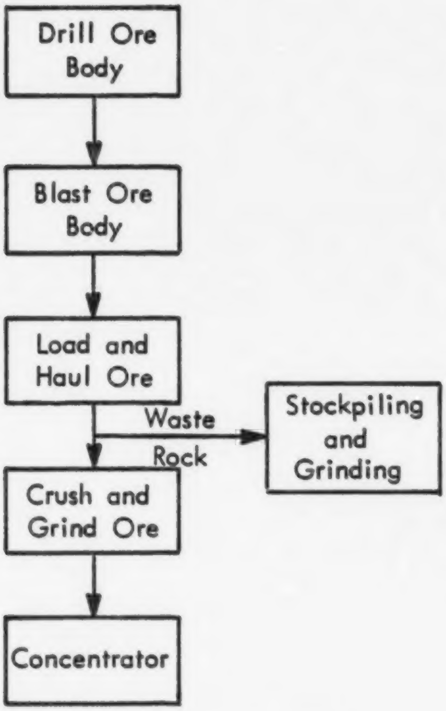
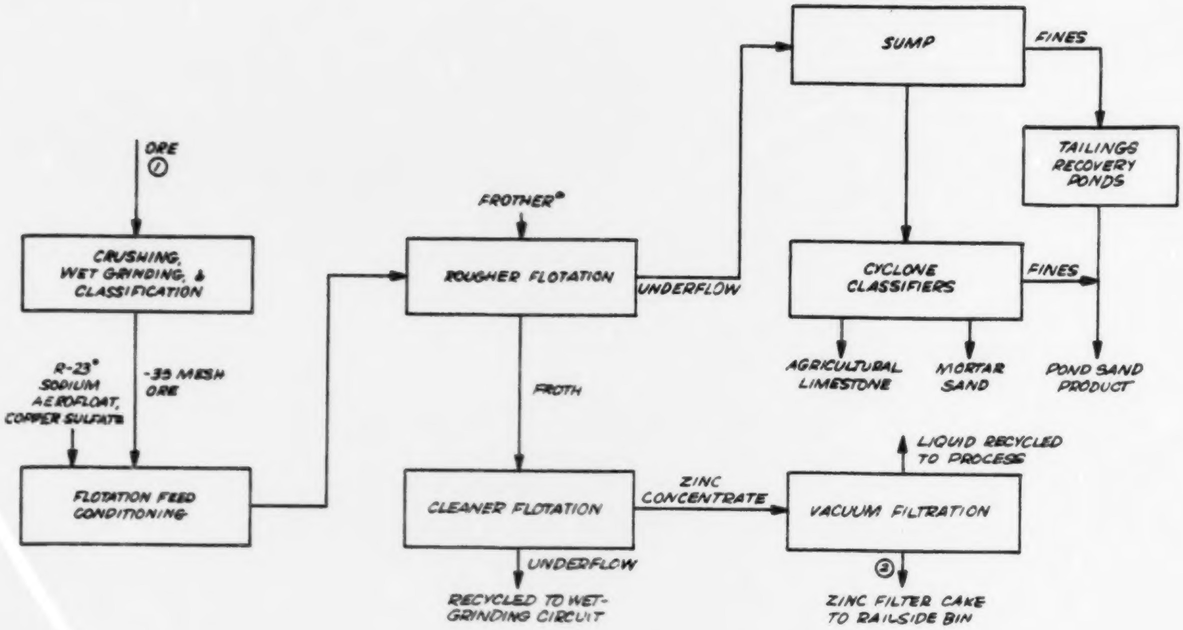
* AN-FO - Ammonium nitrate-fuel oil blasting agent; it generally contains 5 percent fuel oil.


Drill Ore Body
Blast Ore Body
Load and Haul Ore
Waste Rock
Stockpiling and Grinding
Crush and Grind Ore
Concentrator

Source: Reference 6.

Figure 20. Mining of zinc ore.


ORE
①
CRUSHING, WET GRINDING, & CLASSIFICATION
R-23* SODIUM AEROFLOAT, COPPER SULFATE
-35 MESH ORE
FLOTATION FEED CONDITIONING
FROTHER*
ROUGHER FLOTATION
UNDERFLOW
FROTH
CLEANER FLOTATION
UNDERFLOW
RECYCLED TO WET-GRINDING CIRCUIT
ZINC CONCENTRATE
VACUUM FILTRATION
LIQUID RECYCLED TO PROCESS
②
ZINC FILTER CAKE TO RAILSIDE BIN
SUMP
FINES
TAILINGS RECOVERY PONDS
CYCLONE CLASSIFIERS
FINES
AGRICULTURAL LIMESTONE
MORTAR SAND
POND SAND PRODUCT

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS		① CRUDE ORE	② ZINC CONCEN-TRATE
Quantity,	TPY	500,000	30,000
	Metric TPY	453,593	27,216
Assay, %	Zn	4.07	64.00
	Cd	Present	0.25
	Pb	0	0.00
	Cu	0	0.00
	S	Present	32.00
	Unspecified	NA	3.75
	Limestone	Present	NA
	Dolomite	Present	NA

*There are no waste materials in this plant other than liquid discharged to tailings recovery ponds. All solids are sold as commercial products.

FLOTATION ADDITIVES*

NAME	LB/TON OF ORE	G/MT OF ORE
Copper Sulfate	0.5	250
R-23 (Dow Chemical)	0.08	40
Frother 65*	0.02	10
Sodium Aerofloat*	0.07	35

Source: Reference 7.
* Produced by American Cyanamid.

Figure 21. Mining and concentrating of zinc.

Crude ore delivered from the mine is crushed and wet-ground to -35 mesh. Crushing is done by a jaw crusher, a gyratory crusher, and secondary cone crushers. Grinding is carried out by using rod mills in closed circuit with ball mills. The ground ore is classified by a cyclone, with the oversize returned to the ball mills, and the undersize (-35 mesh) material sent to flotation.

Ground ore is conditioned by incorporation of flotation additives, and then floated in banks of flotation cells and cleaner cells. The quantities of flotation additives used are shown in Figure 21. Zinc concentrate discharged by the cleaner cells is vacuum filtered to produce a zinc filter cake which is stored or shipped by rail to a smelter.

Underflow from the flotation machines is passed to a sump, and dewatered and classified as to coarseness by cyclones to yield mortar sand and agricultural limestone products. Slurry from the sump is discharged to tailings recovery ponds. All of the tailings sand is reclaimed from these ponds and sold for commercial uses.

As shown in Figure 21, for the example process, 27,216 MT/year (30,000 tons/year) of 64 percent zinc concentrate are produced from 454,000 MT (500,000 tons) of zinc ore containing 4.07 percent zinc. The product which is shipped to a smelter also contains 0.25 percent cadmium, a potentially hazardous material. This cadmium is recovered at the refinery. The by-products include ground rock formed from waste rock, agricultural limestone, mortar sand, and pond sand produced by classifying the flotation tailings.

The concentrator wastes (tailings) consist of finely divided sands (-35 mesh). These wastes contain a small amount of zinc, along with large amounts of limestone and dolomite. Since the feed ore contains a cadmium compound, a potentially hazardous substance, the tailings may also contain trace amounts of this material. However, no analytical data are available to establish the presence of cadmium in this waste. Based on the information available to the investigators, it was concluded that zinc flotation tailings are not potentially hazardous.

Waste Treatment and Disposal

There are no potentially hazardous wastes resulting from the treatment and disposal of wastes generated in the mining and concentrating of zinc ores. This conclusion was reached as a result of the site visits.

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SECTION IV

MERCURY ORES (SIC 1092)

Industry Characterization

History of the Industry. The first recorded mention of mercury, also known as quicksilver, was by the Greek philosopher Aristotle in the fourth century B.C., when it was used in religious ceremonies. Until the 16th century consumption was small and chiefly for medicinal and cosmetic purposes.

Following the introduction in 1557 of the Patio amalgamation process for recovery of silver, large quantities of mercury were used for amalgamation purposes in Mexico, Peru, and other countries. The invention of the barometer in 1643 introduced mercury into scientific research, and in 1720 Fahrenheit invented the mercury thermometer.

The early history of mercury in the United States is closely associated with the discovery of gold and the development of gold mining in California. Output of mercury began shortly before 1850 in the United States with the advent of the "gold rush" to California in 1849.

Until World War I, the largest and principal use of mercury in the United States was in the amalgamation process of recovery of gold. Significant quantities of mercury were used also in fulminate, drugs, and antifouling paint.

In 1944, production began on the mercury dry-cell battery, and the electrical apparatus category became the principal use for mercury. The mercury-cell process to produce caustic soda and chlorine became widespread following World War II, and a number of such plants have been installed in the United States. In recent years, the relighting of many streets with mercury lamps, the production of hearing-aid batteries, and other uses in electrical equipment have contributed to the high usage in this category.

The mercury industry throughout the world is comparatively small in terms of quantity produced, value of production, and number of producers. Major mercury producing countries include the Soviet Union, Spain, Italy, Mexico, Canada, Peoples Republic of China, and Yugoslavia.

Domestic Production and Capacities. In 1972, there were 37 producing mines in the United States, down from 56 mines in 1971. The number of producing mines dropped to 24 in 1973, and to two mines in November of 1974. Table 66 gives the amount of mercury ore treated in the United States by year.

TABLE 66

MERCURY ORE TREATED IN THE UNITED STATES*

Year	Ore treated (metric tons)
1969	392,440
1970	385,109
1971	241,121
1972	74,915†
1973	23,820

Source: Reference 1.
* Revised.
† Excludes mercury produced from old surface ores, dumps, and as a by-product.

Production of mercury was 251 MT (7,286 flasks) in 1972, down from 942 MT (27,296 flasks) in 1970 (Table 67). These figures were further reduced in 1973 when only 76 MT (2,200 flasks) were produced. Although data are not available for 1974, it is assumed that mercury production would be considerably below the 1973 production level. These declining trends in the mercury industry are a direct result of restrictions in the use of mercury in cosmetics, and EPA standards for plants producing and using mercury and its compounds. With the decline in the U.S. primary production, the nation will be dependent on foreign sources for its supply. This dependence increased from 72 percent of primary consumption in 1972 to almost 100 percent in 1974. Demand for mercury is expected to increase at an annual rate of less than 1 percent through 1983.

TABLE 67

MERCURY PRODUCTION - 1968-1973
(METRIC TONS)*

Year	United States	Rest of world	Total
1968	996	7,963	8,959
1969	1,023	8,957	9,980
1970	942	8,856	9,798
1971	617	9,683	10,300
1972	251	9,391	9,642
1973	76	9,246	9,322
1974†	69	--	--

Source: Reference 2.
* Calculated to metric tons.
† Estimated.

Number, Location, Size, and Age of Mines and Mills. In 1974, there were 10 mercury mines and mills in operation, but all except two of them shut down during the year (Table 68). Both of these operations are open-pit mines located in Napa County, California (Figure 22). While production data are not available, the two mines are very small, with an estimated employment of about 25 workers for the combined operations. Both mines are relatively new, having been established since 1967. However, in both cases there were underground mercury mines located on the site, which operated for a number of years and had been closed down for at least 10 years.

Employment. The two active mercury mines employed fewer than 50 workers in 1974.

By-Products and Coproducts. Domestic zinc producers recover coproduct mercury from smelter operations. In 1974, mercury mines produced no by-products of value in the mining operation.

Waste Generation and Characterization

Waste Generation. There are two main sources of waste from the mining of mercury ores: overburden, and waste rock. Overburden, composed of soil, gravel, clay, and rock, is the material covering ore deposits which is stripped during surface mining operations. Waste rock, usually containing chert, iron pyrite, and sulfur is the "non-ore" refuse material associated with the ore mined.

TABLE 68

NUMBER, LOCATION, AND SIZE OF ACTIVE MERCURY MINING AND MILLING OPERATIONS (1974)

State	Employees (1-19)	Total operations	No. of mills	Total No. of mines
California	2	2	2	2
U.S. total	2	2	2	2
EPA regions				
Region IX	2	2		2

Source: Reference 3.

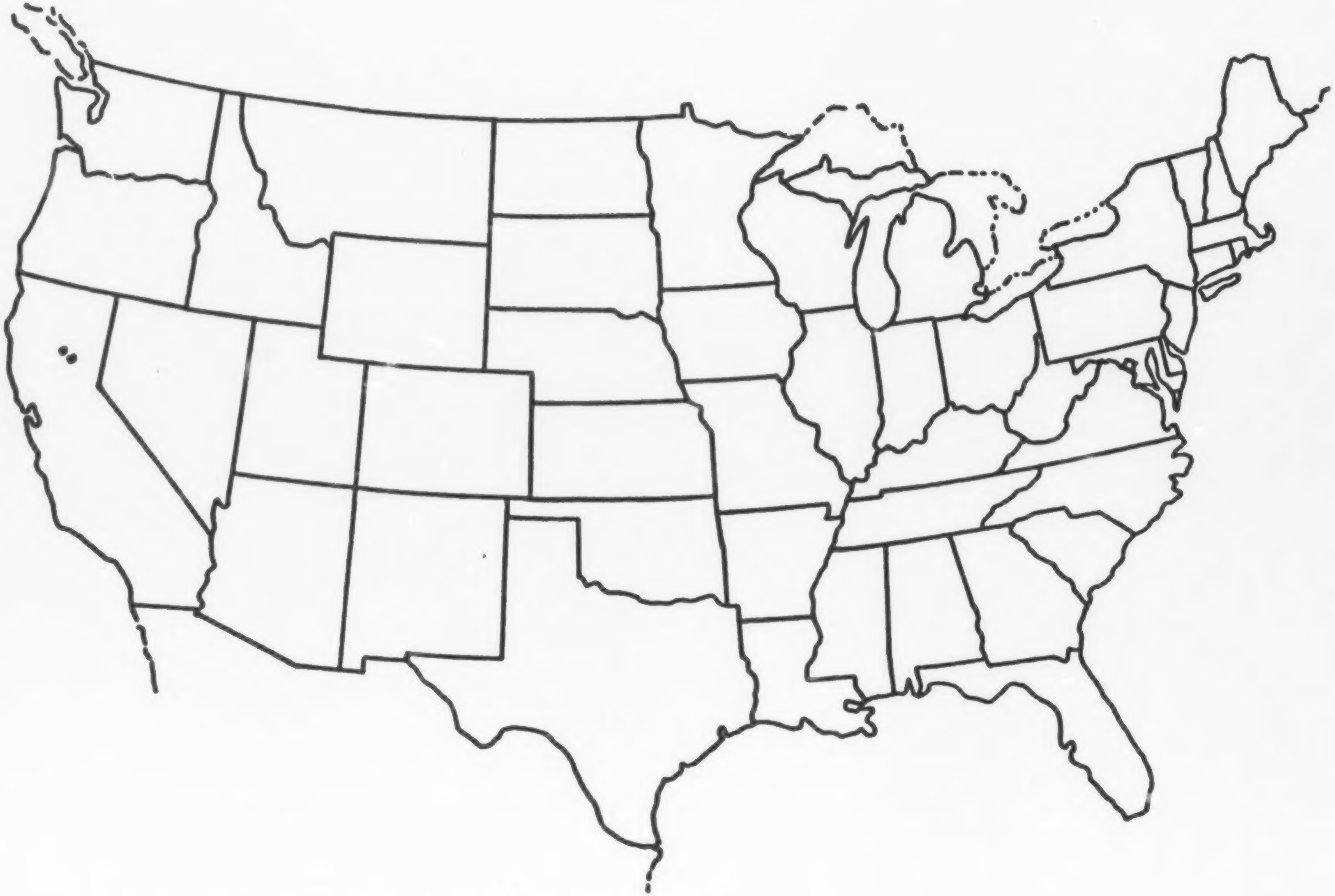


Figure 22. Location of mercury (SIC 1092) mines for 1974.

Table 69 contains the production data for mercury mining for 1974. The scale of operation of mercury mines in the United States ranged from about 5,000 to 12,000 MT (5,512-13,228 tons) of ore mined per year per mine. The two operating mines accounted for the production of 22,000 MT (24,000 tons) of ore in 1974. The amount of waste generated from the mining of mercury ore at these mines in 1974 was 1,416,000 MT (1,560,000 tons), consisting of 908,000 MT (1 million tons) of waste rock and 487,000 MT (537,000 million tons) of overburden. Table 70 also contains the data for ratio of waste to ore. The amount of waste associated with the two mines was very similar, with ratios of waste rock-to-ore and overburden-to-ore of about 41:1 and 22:1, respectively.

Mining Processes.

Open-Pit Mining.

Description of a Typical Process. A flow sheet for mercury ore open-pit mining operations is shown in Figure 23. At the time we started our mine visits and receipt of questionnaires, there were only two operating mercury mines in the United States. Both of these mines were originally underground mines, with the initial mining taking place over 100 years ago. The mines have been converted within the last 10 years to open-pit mines and are less than 80 m (262 ft) deep.

Conventional earth-moving equipment is used at both mines, and the overall mining methods are very similar. The mines drill and blast to loosen overburden and waste rock. Ore is loaded by shovel into trucks and hauled to a concentrator located on the site.

The waste from mercury mining consists of waste rock and overburden. These materials are generally mixed and used in land reclamation for filling gulleys near the two operating mines. One facility leaves part of the waste in the pit where it is being mined. They also stockpile top soil for use as a cover over the waste dump to aid in revegetation.

Groundwater intrusion is an operating problem in these mines. Mine water is pumped out of the mines and discharged to nearby streams. Both mines operate only during the dry season, usually from April to November, and stop mining when the rainy season begins.

Mass Balance of Materials. The mass balance for mercury mines shows that 1,395,000 MT (1,537,724 tons) of overburden and rock are mined to recover 22,000 MT (24,000 tons) of ore.

Description of Individual Waste Streams. The waste streams from the open-pit mining of mercury consist of overburden and waste rock. The overburden is comprised of soil, sandstone, and clay. The waste rock varies widely in composition, but usually contains chert, and may contain iron pyrite, and sulfur. In most cases, waste materials contain sufficient plant nutrients to support vegetation on the surface of the waste piles.

TABLE 69

PRODUCTION STATISTICS BY STATE AND EPA REGION, SIC 1092, FOR MERCURY (METRIC)

			Wastes (10^3 TPY)				Products (TPY)				Ratio of			
State	Region	Ore mined (10^3 TPY)	Waste rock	Overburden	Concentrator wastes	Total waste	Primary metal	By-product metal	Miscel-laneous	Total products	Total waste to ore	Total waste to product	% Total waste	Metal ore
California	IX	22	908	487	21	1,416	69	0	0	69	64	20,000	-	Hg

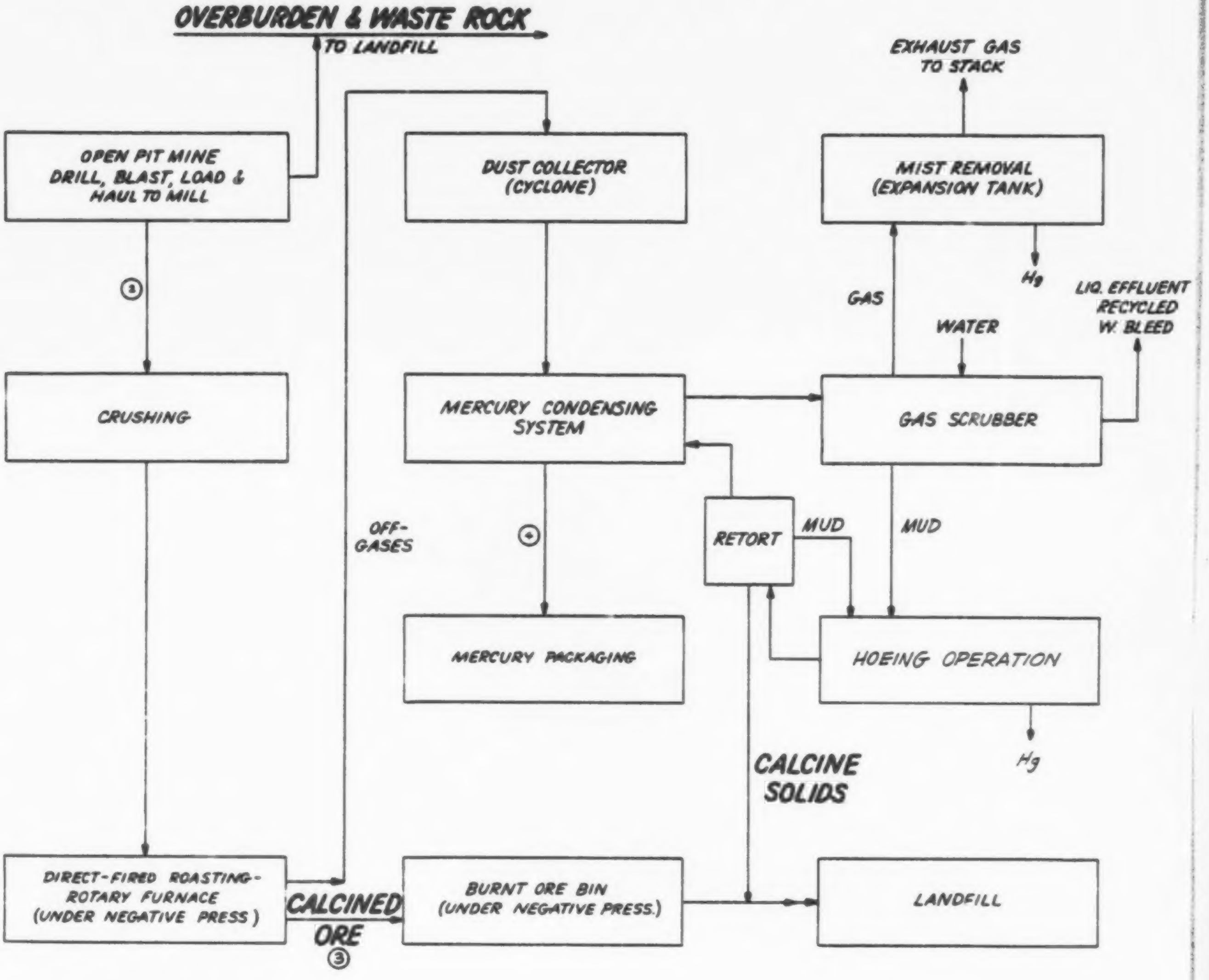
Source: Reference 5.

TABLE 70

RATIO OF TOTAL WASTE ROCK--OVERBURDEN--CONCENTRATOR WASTE TO ORE MINED, SIC 1092, FOR MERCURY ORES (METRIC)*

State	Region	Ore (10^3 TPY)	Total waste rock (10^3 TPY)	Ratio of total waste rock to ore	Overburden (10^3 TPY)	Ratio of total overburden to ore	Tailings (10^3 TPY)	Ratio of tailings to ore
California	IX	22	908	41	487	22	21	0.95

* Compiled from data in Table 69.


OVERBURDEN & WASTE ROCK
TO LANDFILL
EXHAUST GAS TO STACK
OPEN PIT MINE DRILL, BLAST, LOAD & HAUL TO MILL
DUST COLLECTOR (CYCLONE)
MIST REMOVAL (EXPANSION TANK)
Hg
GAS
WATER
LIQ. EFFLUENT RECYCLED W. BLEED
①
CRUSHING
MERCURY CONDENSING SYSTEM
GAS SCRUBBER
OFF-GASES
④
RETORT
MUD
MUD
MERCURY PACKAGING
HOEING OPERATION
Hg
CALCINE SOLIDS
DIRECT-FIRED ROASTING-ROTARY FURNACE (UNDER NEGATIVE PRESS)
CALCINED ORE
③
BURNT ORE BIN (UNDER NEGATIVE PRESS.)
LANDFILL

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEM	① OVERBURDEN & WASTE ROCK	② ORE	③ CALCINED ORE	④ MERCURY
Quantity, TPY	1,538,000	24,000	23,000	76
Metric TPY	1,395,000	22,000	21,000	69
Assay, Hg %		0.615	0	100
FeS_2 %		1.8	0	
Fe %			1	

Source: Reference 5.

Figure 23. Mercury mining and concentrating process.

Identification of Potentially Hazardous Waste Streams. Due to the small scale of the mercury mining industry in the United States, which involves negligible amounts of hazardous material in the mining waste, it is concluded that no potentially hazardous wastes are generated by the domestic mining of mercury ore.

In common with all open-pit mining operations, the inactive surface dumps require protection by a cover of vegetation in order to prevent erosion by rain and wind.

Concentrator Process.

Description of Process. All mercury presently produced in the United States is recovered by heating the mercury ore in furnaces and retorts. This roasting process is an efficient method for distilling the mercury from the ore.

A typical flow diagram for the concentrating of mercury ore is shown in Figure 23. A discussion of the operating steps is given in the following subsection.

Crushing. Ore is hauled from the mine to the mill where it is either spread on the ground for sorting and drying or placed directly in storage bins. The mercury ore is crushed to about 2.5 cm (1 in.) in size by jaw crushers. One mine has a screening operation, and over-sized particles are returned to the crushing operations. The fines are stored in a fine ore bin and then fed continuously by a feeder into a roasting furnance. One of the mines has no covered storage, and ore that is mined must be processed within a day or two. When the rainy season starts, the entire operation closes down.

Roasting. Mercury ore is fed continuously into a direct-fired rotary furnace where it is heated to a temperature of about 750 C by combustion gases. The furnace is oil fired using a Harick burner and is fired countercurrently with flame impingement on the ore. The mercury vaporizes at about 443 C. The cinnabar is oxidized into mercury and sulfur dioxide, and these gases are withdrawn by an exhaust fan along with the combustion gases.

Calcined ore is discharged into a burnt-ore bin where it is held for a soaking period and then batch-discharged into trucks and hauled to waste piles. The furnace and burnt-ore bin are kept under negative pressure to prevent escape of any gases.

Cyclone. The mercury and other exhaust gases are passed through a cyclone separator where dust is removed.

Condensing System. After dust is removed, the gases are passed through a series of 40-cm (16-in.) cast iron pipes connected at the top and bottom with "U" type bends and hoppers. Here the gases are cooled and condensed to form a liquid. The condenser hoppers are equipped with spouts which extend into water-immersed mercury collection pots. The water acts as a seal to prevent the escape of any mercury vapors. Here, mercury is recovered and packaged into flasks.

Gas Scrubber. The gas leaving the condensing system is processed through a Venturi scrubber to remove particulates and dissolved SO_2, and to cool the gas and condense any remaining mercury vapor so that the gases can be emitted to the atmosphere. The gas leaving the scrubber passes into an expansion tank for mist removal. The gas from the expansion tank is emitted to the atmosphere. The gas leaving the scrubber passes into an expansion tank for mist removal. The gas from the expansion tank is emitted to the atmosphere.

Hoeing. Sludges from the gas scrubber are processed by rake and hoeing operations to recover metallic mercury.

Retort. After the metallic mercury is removed from the sludges by hoeing, the remaining sludges are retorted to recover any mercury still present. The retorting of the sludges is a batch operation, a very small operation compared to the continuous furnace roasting operations. The gaseous product of the retorts is passed through the condensing system, and the calcined solids are removed.

Mass Balance of Materials. Figure 23 shows a material balance for one mercury concentrating plant. There are no chemicals added in mercury concentrating.

Description of Individual Waste Streams. The only waste streams from the concentrating of mercury are the calcined ore from the furnace and retort operations.

Identification of Potentially Hazardous Waste. Mercury concentration waste consists of calcined material that contains no hazardous materials. It is concluded that no potentially hazardous waste is generated by the domestic concentrating of mercury ore.

Waste Treatment and Disposal

No potentially hazardous wastes result from the treatment and disposal of wastes generated in the mining and concentrating of mercury ores at the only two operating facilities. This conclusion was reached as a result of the site visits.

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4. Unpublished company data.

5. MRI communication with mercury mining companies.

SECTION V

URANIUM-RADIUM-VANADIUM ORES (SIC 1094)

Industry Characterization

History of the Industry.

Uranium. Uranium and vanadium are frequently found in the same mineral ore, and radium has always been obtained from uranium ores, in which it is formed as a radioactive decay product. The uranium-radium-vanadium ores are, therefore, grouped into one industrial segment.

Uranium was discovered in 1789 by Marten Klaproth in pitchblende from a mine in Germany.[1/] The element was first isolated in 1842.

Radioactivity was discovered in 1896, and radium, a daughter of uranium decay, was discovered by the Curies and Bemont in 1898 in pitchblende from Czechoslovakia.[1/] In the early 1900's radium became important in medical therapy; this led to the search for uranium ores as a source of radium.[1/] The first important sources of radium outside Czechoslovakia were the uranium-vanadium sandstone deposits in western Colorado and eastern Utah from which about 275,000 MT (303,137 tons) of ore were produced during 1898 to 1923.[1/] This ore yielded about 200 g (0.44 lb) of radium, 2,000 MT (2,205 tons) of vanadium, and a small amount of uranium; most of the uranium went into the tailings.[1/]

In 1936, mining of the uranium-vanadium ores increased markedly because of increased demand for vanadium.[1/] Carnotite ores were mined from about 1915 until World War II for recovery of their vanadium content. Since World War II, the ores have been mined for both the uranium and the vanadium values.

Before the discovery of fission in 1939, the known uses for uranium and its compounds were relatively unimportant. The first plant to treat uranium ore was built in Czechoslovakia in 1906; the product sought was radium, not uranium. A plant was built in Denver, Colorado, in 1913 to process handpicked carnotite ore from the Colorado plateau; the radium was recovered in this plant, and most of the uranium was wasted.[2/]

Uranium did not achieve prominence until the successful development of the "atom bomb" during World War II.[2/] Before this, interest in uranium ore centered on the radium content, and much of the early work on radium was done by Madame Curie.[2/]

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The United States remained the major world producer of radium from 1911 to 1923.[2] From about 1912 until the early 1940's, the uranium industry was small, and the uses for carnotite ores were few. The requirements for uranium were dramatically changed by events during World War II. Military developments in the early 1940's demanded large quantities of uranium, and ores from the Belgian Congo, the Republic of South Africa, Australia, and Portugal were imported. Prior to 1946, annual domestic production of uranium was small and did not reach 50 percent of procurement until 1953. The purpose of nuclear developments up to 1945 remained military, even though the peacetime possibilities of nuclear energy became apparent when the first chain reaction was confirmed. In 1946, the Atomic Energy Commission was established, and it was given the full burden of nuclear responsibility formerly held by the Army's Manhattan District. As an independent civilian agency of the Federal Government, the Atomic Energy Commission (AEC) in 1946 assumed the responsibility for the development, use, and control of atomic energy for the defense and security of the United States and for peaceful applications.[2] Within recent years, a new federal agency, the Energy Research and Development Administration, has assumed the functions of the AEC.

There has been no domestic production of radium since 1950.[3] The small domestic demand was met by imports, withdrawal from dealers' stocks, and reprocessing. Radium Chemical Company, Inc., New York, was the main dealer in the United States.[3] During 1974, radium was used mostly in therapeutic treatment of cancer; however, in medical and industrial applications, cheaper and less hazardous isotopes have continued to replace radium.[3] For these reasons, radium processing is not discussed in this report.

The uranium industry in the United States consists of many privately-owned firms engaged in exploring, drilling, mining, milling, fabricating, and chemical processing of uranium.[2]

Major free world producers of uranium include the United States, Canada, Republic of South Africa, Niger, France, and Gabon.[2]

Vanadium. Vanadium, when discovered in Mexico in 1801, was thought to be a form of chromium. The metal was not recognized as a new element until 1831. The vanadium content of the earth's crust has been estimated at about 1,500 g/MT (48 oz/ton)--more than copper, lead, or zinc. Vanadium, however, is widely distributed throughout a great variety of rocks and does not tend to concentrate and form ore deposits as readily as do the base metals. Recovery is principally as a coproduct of other metals; rarely are deposits mined for the vanadium content alone. One notable exception in the United States is the vanadium mine near Hot Springs, Arkansas.

The first commercial use for vanadium developed about 1860 as vanadium salts for making ink and in coloring fabrics, leather, glass, and pottery. Its use as an alloying element in steels began about 1905 by the U.S. automobile industry. However, its beneficial effects in armor plate, cutlery, tools, and constructional steels were recognized in France and England as early as 1896. The manufacture of specialty steels has continued to supply the principal market for vanadium. Vanadium has important catalytic properties, but this application has remained small. Since 1960, the use of vanadium in nonferrous alloys for jet engines and airframes has become increasingly important. Major world producers of vanadium include the Republic of South Africa, U.S.S.R., the United States, Finland, and Norway.

Domestic Production and Capacities.

Uranium. Production data, by state, for domestic mining of uranium ore are shown in Table 71. These data show the recoverable contents of U_3O_8 in the mined ore. During the period of 1970 through 1974, New Mexico and Wyoming were the dominant producing states.

In 1970, the distribution of total mine output was 52.9 percent (6,061 MT of U_3O_8 from underground mines, 44.8 percent (5,133 MT of U_3O_8) from open pits, and 2.3 percent (264 MT of U_3O_8) from other sources (leaching, mine water treatment, and raffinate).[13]

During 1971, underground mines accounted for 45.3 percent (5,108 MT of U_3O_8) of total mine output, open-pit mines produced 53.3 percent (6,009 MT of U_3O_8), and miscellaneous activities provided the remaining 1.4 percent (158 MT of U_3O_8).[14]

In 1972, 40 percent (4,768 MT of U_3O_8) of mining output was from underground mines, 58 percent (6,914 MT of U_3O_8) was from open pits, and 2 percent (238 MT of U_3O_8) was from miscellaneous operations.[10]

During 1973, open-pit mines contributed 62.5 percent (7,423 MT of U_3O_8) of the total mine output, while 36.1 percent (4,287 MT of U_3O_8) came from open-pit mines, and miscellaneous operations (recovery of U_3O_8 from mine waters, heap leaching, solution mining, and raffinate) represented 1.4 percent (166 MT of U_3O_8) of the total output.[5]

Mining production during 1974 amounted to 11,430 MT (12,600 tons) of U_3O_8,[6] underground mining contributed 40 percent (4,664 MT of U_3O_8) of total mine output, open-pit mines accounted for 58 percent (6,762 MT of U_3O_8), and 2 percent (233 MT of U_3O_8) was from miscellaneous operations (heap leaching, mine water treatment, solution mining, and low-grade stockpiles).[11]

TABLE 71

URANIUM ORE MINING PRODUCTION IN THE UNITED STATES, BY STATE[*], [†], [‡]
(RECOVERABLE CONTENT U_3O_8 METRIC TONS)

	1970[§]	1971[§]	1972[§]	1973[§]	1974
Colorado	1,237	1,150	851	871	-
New Mexico	5,249	4,792	4,902	4,145	4,899[††]
Utah	741	655	678	880	-
Wyoming	2,878	3,168	3,875	4,562	3,719[††]
Other states	1,089	1,352[†]	1,376[†]	1,252[†]	2,812[††],[*]
Miscellaneous sources all states[‡‡]	264	158	238	166	229
Total	11,458	11,275	11,920	11,876	11,659
Average ore grade, % U_3O_8[†]	0.202[†]	0.205[†]	0.213[†]	0.208[†]	0.2[‡]

[*] The other states include Colorado, Utah, Washington, and Texas.

[†] Source: Reference 5. Includes Alaska, South Dakota, Texas, and Washington, in 1971 and 1972. The other states for 1973 include Alaska, Texas, and Washington.

[‡] MRI estimate based on listing of average grade from 1970 to 1973 and responses to MRI written questionnaires.

[§] Source: Reference 4. Data are based on concentrator recovery factors.

[††] Source: Reference 6.

[‡‡] Includes uranium recoverable in miscellaneous operations (treatment of mine waters, solution mining, heap leaching, and raffinate).

Domestic U_3O_8 production is expanding to meet growing uranium fuel demand. Nuclear power development, delayed by environmental opposition and other sources, is expected to be expedited because of the energy crisis.[7]

Salient statistics for domestic production of uranium concentrate from 1969 through 1974 are presented in Table 72. The annual production of ore increased from 5,356,000 MT (5,904,000 tons) in 1969 to 5,930,000 MT (6,536,700 tons) in 1973; the average grade of ore during this period varied from 0.20 to 0.213 percent U_3O_8.

Annual domestic production of uranium concentrate (U_3O_8) during the period 1969 through 1974 ranged from 10,531 to 12,007 MT (11,600-13,240 tons).

Capacity data for domestic concentrator plants which were operated during the period of 1969 through 1974 are shown in Table 73. During this period, the number of active concentrator plants varied from 15 to 19, and the total nominal capacity ranged from 21,591 MT/day (23,800 tons/day) of ore in 1969 to 28,941 MT/day (31,900 tons/day) of ore in 1972. In 1974, the processing capacity of the concentrator plants ranged from 408 to 6,350 MT (450-7,000 tons) of ore per day.

The utilization of total concentrator production capacity was 69 percent during 1972 and 1973 and 65 percent in 1974.

As shown in Table 72, the total amount of U_3O_8 concentrate produced domestically in 1973 was 12,007 MT (13,236 tons) of U_3O_8. Demand for uranium concentrate is expected to increase at an average annual rate of 18 percent through 1980.[7] MRI estimates that this 18 percent annual increase will continue from 1980 through 1983. At this rate of increase in demand, U_3O_8 production could reach 23,280 MT (25,660 tons) of U_3O_8 in 1977 and 62,840 MT (69,270 tons) of U_3O_8 in 1983.

The AEC has terminated its role as a lessor of enriched uranium to industry. All government-owned special nuclear materials (leased from the AEC for use in facilities licensed under Sections 103 and 104b of the Atomic Energy Act) were returned to the AEC or converted to private ownership before July 1973.

The present mining and concentrating sectors are dominated by subsidiaries or affiliates of major mining and petroleum companies. Many operations are integrated, and the trend is toward further vertical integration into uranium refining, enrichment, fuels manufacture, and reprocessing of spent nuclear fuels.

TABLE 72

SALIENT URANIUM CONCENTRATE (U_3O_8) STATISTICS FOR THE UNITED STATES*
(METRIC TONS U_3O_8 UNLESS OTHERWISE SPECIFIED)

	1969	1970	1971	1972	1973	1974
Production						
Mine						
Ore (thousand MT)	5,356	5,737	5,696	5,822	5,930	5,715**
U_3O_8 content of ore	11,141	11,583	11,709	12,398	12,327	11,430††
Average grade of ore, % U_3O_8	0.208	0.202	0.205	0.213	0.208	0.20**
Recoverable†,‡	10,768	11,059	11,122	11,684	11,703	-
Mill concentrate§	10,531	11,707	11,134	11,703	12,007	10,614††
Deliveries of concentrate						
Atomic Energy Commission						
Quantity	5,610	2,286	-	-	-	-
Value (thousands)	$72,336	$28,078	-	-	-	-
Price per pound	$5.85	$5.59	-	-	-	-
Private industry††	5,625	8,437	11,612	10,523	10,977	-
Imports, concentrate	1,364	603	855	2,113	5,085	-

* Principal source (exceptions are noted): Reference 5.

† Estimate.

‡ Receipts at mills; excludes uranium from leaching operations, mine waters, and refinery residues.

§ Includes marketable concentrate from leaching operations.

** MRI estimate based on history of average grade from 1969 to 1973.

†† Source: Reference 6.

TABLE 73

ACTIVE URANIUM ORE CONCENTRATOR PLANTS*

State and company	Concentrator location	EPA region	Nominal capacity, metric tons of ore per day* 1969	1970	1971	1972†	1973‡	1974§
New Mexico								
The Anaconda Company	Bluewater	VI	2,722	2,722	2,722	2,722	2,722	2,722
Kerr - McGee Corporation	Grants		5,443	6,350	6,350	6,350	6,350	6,350
United Nuclear - Homestake Partners	Grants		3,175	3,175			3,175	3,175
United Nuclear - Homestake Mining Company	Grants				3,175	3,175		
Texas								
Susquehanna - Western, Inc.	Falls City		907	907	907	907		
Susquehanna - Western, Inc.	Ray Point			907	907	907		
Continental Oil Company - Pioneer Nuclear, Inc.	Karnes County					1,588	1,588	1,588
Subtotal			12,247	14,061	14,061	15,649	13,835	13,835
Colorado								
American Metal Climax, Inc.	Grand Junction	VIII	454					
Cotter Corporation	Canon City		363	408	408	408	408	408
Union Carbide Corporation	Rifle		1,361	1,814	1,814	1,814		
Union Carbide Corporation	Uravan						1,814	1,814
South Dakota								
Mines Development, Inc.	Edgemont		590	590	590	590		
Utah								
Atlas Corporation	Moab		1,361	1,361	1,361	1,361	1,361	1,361
Rio Algom Mines, Ltd.	LaSal					454	454	454
Wyoming								
Federal - American Partners	Gas Hills		816	862			862	862
Petrotomics Company	Shirley Basin		907	1,361	1,361	1,361	1,361	1,361f/
Union Carbide Corporation	Gas Hills		907	907	907	907	907	907
Utah Construction and Mining Companye/	Gas Hills		1,089	1,089	1,089	1,089	1,089	1,089
Utah Construction and Mining Companye/	Shirley Basin				1,089	1,089	1,089	1,089
Federal Resources Corporation - American Nuclear Corporation	Gas Hills				862	862		
Western Nuclear, Inc.	Jeffrey City		1,089	1,089	1,089	1,089	1,089	1,089
Exxon Company	Powder River Basin					1,814	1,814	1,814
Subtotal			8,936	9,481	10,570	12,838	12,248	12,248
Washington								
Dawn Mining Company	Ford	X	408	454	454	454	454	454
Total			21,591	23,996	25,085	28,941	26,537	26,537
No. of concentrator plants			15	15	16	19	16	16

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* Sources: References 5, 6, 8, 9, 10, 12, 13, and 14.

† Utilization of total concentrator capacity in 1972 was 69% (Reference 8).

‡ Utilization of total concentrator capacity in 1973 was 69% (Reference 9).

§ Utilization of total concentrator capacity in 1974 was 65% (Reference 6).

** The company name in 1974 was changed to Utah International, Inc.

†† The Petrotomics concentrator in Shirley Basin, Wyoming, was shut down in 1974, but is included in the table since it is part of a major production center and can be reopened.

Phosphate rock containing U_3O_8 is a potential source of uranium. In the fertilizer industry, the rock is reacted with sulfuric acid to form phosphoric acid, and most of the uranium present goes into the product acid. One of the recovery processes owned by Uranium Recovery Corporation makes a concentrated strip solution that is transported to a central processing plant in Florida.[11] Initial operation of conversion plants for production of U_3O_8 from the solution is expected in 1976.[11]

Vanadium. The principal domestic sources of vanadium are the Colorado Plateau uranium-vanadium ores, the Arkansas vanadium ore, and the phosphate rock in Colorado. These include Union Carbide's mine at Wilson Springs, Arkansas, and Union Carbide's uranium-vanadium properties on the Colorado Plateau. The Wilson Springs, Arkansas, operation is the only one producing vanadium from vanadium ore on a large scale. Table 74 gives the mine production of vanadium produced in the United States by year. In 1973, the production of vanadium ore from this mine was 454,919 MT (501,462 tons). U.S. vanadium production in 1974 is estimated at 8,763 MT (9,660 tons) (recovered vanadium concentrate products) (Table 75).

New sources of vanadium being considered for commercialization include evaluation of a property in southeastern Idaho containing about 20 million metric tons (22 million tons) of ore assaying 1.35 percent V_2O_5.

U.S. consumption of vanadium for all end uses probably reached 6,300 MT (6,930 tons) in 1973, about equal to the 1969 record. The sheer volume of steel production was responsible for the high level of vanadium use; 80 to 85 percent of all vanadium produced is consumed by the steel industry. U.S. demand for vanadium is expected to increase at an annual rate of about 5 percent through 1983. Although domestic reserves are large enough to meet expected demand, cheaper foreign material will probably meet about 25 percent of the demand.

The problem of oversupply of coproduct vanadium from uranium production no longer exists and will probably not recur. The United States can no longer depend on coproduct vanadium from uranium and phosphate mining for an adequate supply. An increase in supply from these sources will be limited by the demand for uranium and phosphorus. To maintain a domestic production capability in vanadium, it will be necessary to (1) discover and develop other deposits such as those at Wilson Springs, Arkansas; (b) exploit the vanadium deposits of the Colorado Plateau and the phosphatic shales of Idaho and Wyoming with little or no recoverable uranium or phosphate; (c) recover vanadium from titaniferous magnetic operations.

TABLE 74

MINE PRODUCTION OF VANADIUM PRODUCED
IN THE UNITED STATES
(MT)

Year	Mine production*
1969	5,205
1970	5,255
1971	5,032
1972	4,263
1973	3,735

Source: Reference 5.

* Measured by receipts of uranium and vanadium ores and concentrates at mills, vanadium content.

TABLE 75

U.S. VANADIUM CONCENTRATE PRODUCTION, 1974*
(RECOVERED VANADIUM CONCENTRATE PRODUCTS)

State	MT
Arkansas	4,536
Colorado	4,227
U.S. total	8,763

EPA regions	MT
Region I	--
Region II	--
Region III	--
Region IV	--
Region V	--
Region VI	4,536
Region VII	--
Region VIII	4,227
Region IX	--
Region X	--

Source: Reference 15.

* Engineering & Mining Journal and unpublished mining company data.

Number, Location, Size, and Age of Mines and Concentrators.

Uranium. Data for uranium mining operations conducted during the period of 1969 through 1974 are presented in Table 76.

The total number of mining properties in the United States declined from 310 in 1969 to 175 in 1973. From 1970 to 1973, the share of total production from open-pit mines increased from 44.8 to 62.5 percent, while the percentage output from underground mines decreased from 52.9 to 36.1 percent. The output from miscellaneous sources, which included recovery of U_3O_8 from mine water, heap leaching, solution leaching, and raffinate, ranged from 1.4 to 2.3 percent of total output.

Estimates for 1974 include 122 underground mines, 33 open-pit mines, and 20 miscellaneous output sources. The majority of the mines are located in New Mexico, Colorado, and Wyoming, with small concentrations in Texas, Utah, and Washington. Data for the number, location, and size of concentrator plants are shown in Table 73.

Uranium mines vary in size from very small operations employing fewer than 20 workers to some that employ more than 500 workers. Most uranium mines are recent developments; practically all have opened since 1950, and most of the active mines in 1974 have opened since 1960. Nearly all of the concentrator plants were placed in operation after 1955; many of the plants were built in the 1960's and the 1970's.

Vanadium. The only mine producing vanadium from vanadium ore on a large scale is located in Wilson Springs, Arkansas. The mine has a production capacity of 762,000 MT/year (839,961 tons/year) and employs about 177 workers.

Employment.

Uranium. Total employment in active uranium mines was 5,202 workers in 1974. Employment was concentrated (81 percent of all uranium mine workers) in New Mexico, Wyoming, and Colorado (Table 77).

Data concerning the total employment for uranium mines and concentrators are presented in Table 78.

Vanadium. In 1974, approximately 177 workers were employed at the only vanadium mine operating in the United States at Wilson Springs, Arkansas.

TABLE 76

DOMESTIC URANIUM MINE OPERATIONS AND PRODUCTION DATA*

Operating year	Total producing properties	Underground mines		Open-pit mines		Other output sources	
		No.	% of Output	No.	% of Output	No.	% of Output
1969	310	-	-	-	-	-	-
1970	263	216	52.9	27	44.8	20	2.3
1971	240	193	45.3	29	53.3	18	1.4
1972	190	141	40.0	37	58.0	12	2.0
1973	175	122	36.1	33	62.5	20	1.4
1974	175b/	122b/	40.0	33b/	58.0	20b/	2.0

* Sources: References 5, 6, 10, 12, 13, and 14.

† Estimated by MRI to be the same as for 1973.

TABLE 77

EMPLOYMENT IN ACTIVE URANIUM MINE OPERATIONS IN 1974*

State	Employment
Colorado	453
New Mexico	2,161
Texas	447
Utah	280
Washington	250
Wyoming	1,611
U.S. total	5,202
EPA regions	
Region I	-
Region II	-
Region III	-
Region IV	-
Region V	-
Region VI	2,608
Region VII	-
Region VIII	2,344
Region IX	-
Region X	250

* Source: Engineering & Mining Journal, 1973-1974. Mining Enforcement and Safety Administration. Unpublished mining company data.

TABLE 78

TOTAL EMPLOYMENT AT DOMESTIC URANIUM MINES AND CONCENTRATORS[7/]

	1969	1970	1971	1972	1973	1974
Employees, for mines and concentrators at year end	9,059	8,165	7,373	6,403	6,500	6,000*

* Estimate by MRI based on data in Reference 15.

By-Product and Coproduct Relationships. A large part of the U.S. vanadium output is a coproduct of uranium at two mills on the Colorado Plateau. Scandium has been recovered in the United States as a by-product of uranium processing, but has not been recovered from domestic residues in recent years. Small quantities of molybdenum occur in the uraniferous lignites of the Dakotas. Copper, zinc, gold, and silver may be recovered during uranium processing.

Waste Characterization

Uranium-Vanadium. Production statistics by state and region for uranium-radium-vanadium ores in 1974 are shown in Table 79. The total ore mined amounted to 5,715,000 MT (6,300,000 tons). The estimated total solid wastes from mining included 93.6 million metric tons of overburden (103 million tons) and 2.2 million metric tons (2.5 million tons) of waste rock. The estimated total dry weight of concentrator wastes amounted to the same quantity as the ore mined and the total wastewater discharged to disposal in tailings ponds was 11.7 million metric tons (12.9 million tons). On a national basis, the ratio of dry weight of total waste to ore was 18 to 1, and the weight ratio of total dry waste to total concentrator products was 7,401 to 1. Region VIII accounted for about 75 percent (75.7 million metric tons) of the total dry wastes and Region VI generated 23 percent (23.9 million metric tons); the remainder of the dry wastes (2 percent (2 million metric tons)) was contributed by Region X.

Data on the ratio of total waste rock, overburden and concentrator waste to ore mined are shown in Table 80. On a national basis in 1974, the weight ratio of waste rock to ore was 0.39, the ratio of overburden to ore was 16.38, and the ratio of concentrator waste (dry solids) to ore was 1.0.

Descriptions of representative domestic processes, mass balances of materials, individual waste streams, and potentially hazardous wastes generated in uranium ore mining and concentrating operations are given in the following subsections.

Open-Pit Mining Waste. Open-pit mines can be operated in conjunction with any type of uranium ore concentrating process. Open-pit mine operations of Wyoming are typical.

Description of Representative Processes. Open-pit, or surface mining, methods are usually applied where the uranium ore deposits are located near the surface and are covered with loosely consolidated soil. In other cases, the choice of method depends basically on the relative mining costs for the required output. The factors which must be evaluated before determining the mining method are the size, shape, grade, depth, and thickness of the ore deposits.[16]

Some mining companies use open-pit methods to depths of more than 152 m (500 ft), but as the depth of overburden above the ore zone increases beyond 91 m (300 ft), underground mining methods usually are more feasible.[16] The equipment used in surface mining varies widely. For mining small deposits to depths of 9 m (about 30 ft), conventional construction-type equipment such as bulldozers, front-end loaders, trucks, gasoline, and diesel-powered shovels are used.[16] For deeper deposits, much larger earth moving equipment is utilized.

TABLE 79

TOTAL PRODUCTION STATISTICS BY STATE AND EPA REGION FOR SIC 1094 (URANIUM-VANADIUM ORES) IN 1974 (METRIC)

State	EPA Region	Ore mined (10^3 TPY)	Waste rock (10^3 TPY)	Over-burden (10^3 TPY)	Concentrator wastes dry weight (10^3 TPY)	Concentrator wastes wet weight (10^3 TPY)	Concentrator products, TPY: Uranium concentrate	Concentrator products, TPY: Vanadium concentrate	Concentrator products, TPY: Other metals	Concentrator products, TPY: Total	Ratio of dry weight waste to ore	Ratio of dry weight waste to products	% Total dry waste in region	% Total wet waste in region
New Mexico		2,449	1,127	12,172	2,449	6,930	5,442	0	39	5,481	6.43	2,873		
Texas		367	484	7,263	367	2,276	706	0	0	706	22.11	11,493		
Total	VI	2,816	1,611	19,435	2,816	9,206	6,148	0	39	6,187	8.47	3,857	23.49	52.96
Colorado		514	0	0	514	1,092	987	1,787	104	2,878	1.00	179		
Utah		420	38	332	420	932	807	0	0	807	1.88	979		
Wyoming		1,860	353	72,224	1,860	5,889	3,649	0	0	3,649	40.02	20,399		
Total	VIII	2,794	391	72,556	2,794	7,913	5,443	1,787	104	7,334	27.11	10,327	74.57	45.53
Washington	X	105	221	1,644	105	263	203	0	0	203	18.76	9,704	1.94	1.51
National		5,715	2,223	93,635	5,715	17,382	11,794	1,787	143	13,724	17.77	7,401	100.00	100.00

Source: Reference 36.

TABLE 80

RATIO OF TOTAL WASTE ROCK--OVERBURDEN--CONCENTRATOR WASTE TO ORE MINED FOR SIC 1094 (URANIUM-VANADIUM ORES) FOR 1974 (METRIC)*

State	EPA Region	Ore (10^3 TPY)	Waste rock (10^3 TPY)	Ratio of total waste rock to ore	Over-burden (10^3 TPY)	Ratio of total O.B. to ore	Concentrate waste (10^3 TPY)	Ratio of total concentrator waste to ore
New Mexico		2,449	1,127	0.46	12,172	4.97	2,449	1.00
Texas		367	484	1.32	7,263	19.79	367	1.00
Total	VI	2,816	1,611	0.57	19,435	6.90	2,816	1.00
Colorado		514	0	0	0	0	514	1.00
Utah		420	38	0.09	332	0.79	420	1.00
Wyoming		1,860	353	0.19	72,224	38.83	1,860	1.00
Total	VIII	2,794	391	0.14	72,556	25.97	2,794	1.00
Washington	X	105	221	2.10	1,644	15.66	105	1.00
National		5,715	2,223	0.39	93,635	16.38	5,715	1.00

* Compiled from data in Table 79.

Since control of ore grade is important, each mine develops a method best suited for its particular ore body. Ore grade is determined by radiation detection equipment. Most mines resort to ore stockpiling and blending; chemical analysis is used to control the blended material.[16]

Stripping of the overburden to expose the top layer of the deposit may be a large part of the cost of mining.[16] The depth of overburden varies from a few meters to 30 m (100 ft) or more.[17] The weight ratio of overburden to uranium ore removed in the mining operations is high compared to other types of mining; the ratios can range from about 0.8:1 to 39:1, as shown in Table 80.

Since uranium mining is carried out principally in low rainfall regions, relatively few mines discharge any water. Wherever feasible, mine water is used as process water in the ore concentrating operations. In these cases, it becomes a component of concentrator effluent. A detailed description of the water balance for domestic uranium mines and concentrators is given in the technical literature.[18]

A representative flow diagram for open-pit mining operations is shown in Figure 24, p. 185. Generally, the mining method consists of drilling, blasting, loading, the broken ore by power shovel into dump trucks, and transporting the ore to a concentrator.

The stripped overburden and waste rock are usually hauled out of the pit and land-disposed on open surface dumps located near the mine. At some mines a portion of the overburden and waste rock is used in mine backfill operations. Mined ore is checked for grade and hauled to a collection area which serves to supply feed ore to a concentrator plant.

Mass Balances of Materials. Mass balance data for a representative model of an open-pit mining operation are shown in Figure 24. The values for mining wastes were calculated from data shown in Table 79, assuming that all mines in Wyoming, Texas, and Washington are open-pit mines and that one plant (The Anaconda Company) in New Mexico operates an open-pit mine. The weighted average weight ratios (overburden to ore and waste rock to ore) for these selected data were calculated and reported for the model mine. The assumed production rate for the model plant was suggested by the literature.[19]

The data (dry weights) in Figure 24 apply for a model open-pit mine which theoretically produces 662,200 MT (730,000 tons) per year of uranium ore (0.20 percent U_3O_8). The corresponding annual quantity of overburden mined is 19,418,000 MT (21,405,000 tons); the weight ratio of overburden to ore is 29 to 1. The annual amount of waste rock mined is 425,700 MT (469,300 tons) and the weight ratio of waste rock to ore is 0.6 to 1.

Industry data presented previously for uranium industry characterization show that in 1974, open-pit mines accounted for about 58 percent of total mine production in terms of U_3O_8 values. The estimated ore production from open-pit mines in 1974 is 3,152,000 MT (3,470,000 tons). The data for dry wastes in Table 79 indicate that a total of 93,635,000 MT (103,215,000 tons) of overburden was generated in 1974; all of this waste is produced by open-pit mining. The estimated total quantity of waste rock mined by open-pit operations during 1974 is 2,026,000 MT (2,233,000 tons). The estimated values are based on the assumptions and data described in the two preceding paragraphs and the data in Table 79.

Only very meager data were obtained concerning the moisture contents of the open-pit mined ore; the moisture content ranges from about 6 to 10 percent.[20] No information was obtained concerning the moisture content of overburden and waste rock.

Description of Individual Waste Streams. The waste streams from open-pit uranium mines are overburden material and waste rock. The overburden generally consists of particles of soil, sandstone, clay, and shale. The waste rock can vary widely in composition, but usually contains chert, feldspar, quartz, and small amounts of radioactive minerals (e.g., natural uranium and its decay products). Usually these waste materials contain sufficient plant nutrients to support some vegetation on the surface of the waste piles.

Identification of Potentially Hazardous Waste Materials. The only potentially hazardous waste generated in open-pit mining is the waste rock, which contains above background levels of radioactive materials (natural uranium and its decay products). All waste rock is considered to be potentially hazardous. The decay products of uranium include several potentially hazardous radioactive substances such as radium, radon gas, and thorium.[19] The total potentially hazardous waste generated by domestic open-pit mines in 1974 consisted of about 2,026,000 MT (2,233,000 tons) of waste rock.

Underground Mining Wastes. Underground mines can be used with any type of uranium ore concentrating process. Underground mines of New Mexico are typical.

Descriptions of Representative Processes. Since the uranium ore deposits vary widely in shape, size, altitude, and grade, underground mines use a variety of mining methods.[16]

In the Colorado Plateau area many mines are simple adits or inclined entries made in a canyon wall or sloping ground to recover small deposits of ore.[16] The small ore bodies are usually mined by open-cast methods, which may be unsupported or supported by roof bolting, and in wide areas pillar supports may be used. In the small mines transportation equipment varies from hand-tramming to mechanical haulage.[16]

The larger and deeper ore deposits require more complex mining methods.[16] The mining techniques include the room-and-pillar, long wall retreat, and panel methods, along with modifications of each of these procedures. The haulageways accommodate either electric locomotives on tracks or large, trackless haulage equipment.[16]

A representative flow diagram for underground mining is shown in Figure 25, p. 191. Generally, the mining method consists of drilling, blasting, moving broken ore to the surface and transporting the ore to a concentrator.

Mine ventilation is a problem, because of the presence of radon gas and its radiation hazard.[16] The decay products of uranium contained in the ore and waste rock include radon gas and radon daughters. Bodily harm (e.g., lung cancer), can occur from inhalation of radon gas and radon daughter products which can attach themselves to dust particles in the air.[21] Significant improvement has been achieved in the mine environment through increased mine ventilation and better distribution of fresh air to all working areas.[16] Fans are used to circulate the air, and large diameter vent shafts are installed to provide sufficient fresh air for the mine.[17]

Groundwater pumped from the mine to the surface is commonly used as processing water in the mill. This water frequently contains significant amounts of dissolved uranium values.

Some companies also conduct special mining operations such as solution mining, heap leaching, and treatment of mine water to recover the uranium values. These special mine production activities accounted for about 2 percent of the total mine output of U_3O_8 values in 1974.

Mass Balances of Materials. Mass balance data which apply for a representative model of an underground mining operation are shown in Figure 25. These values were calculated on the basis of data shown in Table 79, assuming all mining operations in Colorado, Utah and all but one plant (Anaconda Company) in New Mexico are typical of underground mining. The weighted average ratio of waste rock to ore for these selected data was calculated and reported for the model mine. The assumed production rate was suggested by the literature.[19] These data (dry weights) apply for a model mine which conceptually produces 662,200 MT (730,000 tons) per year of crude ore (0.20 percent U_3O_8). The corresponding annual amount of waste rock mined is 51,000 MT (56,200 tons) per year. The weight ratio of waste rock ore is 0.077 to 1 for the model mine; for individual operating mines the ratio can range from 0 to 1 to about 0.1 to 1.

Data indicate that in 1974, underground mines contributed about 40 percent of total mine output in terms of U_3O_8 values. Mining production statistics by state and region for uranium-vanadium ores in 1974 are given in Table 79. For the operations during 1974, the estimated total ore production by underground mines was 2,563,000 MT (2,825,000 tons). The corresponding estimated total quantity of waste rock mined from underground mines was 197,000 MT (217,000 tons). These estimates are based on data in the preceding paragraph and the data in Table 79.

Only very limited data were found concerning the moisture contents of ore recovered from underground mines; the moisture content varies from about 8 to 20 percent.[20]

Description of Individual Waste Streams. The amount of solid waste (waste rock) generated in underground mining operations is small in proportion to ore mines. In several mining operations the waste rock is stored in surface piles for future use as a low-grade uranium ore. The waste rock contains small amounts of natural uranium and its radioactive decay products.

Identification of Potentially Hazardous Waste Materials. All of the waste rock mined in underground mining operations is potentially hazardous because it contains above background levels of radioactive substances such as uranium, radium, and thorium.

In 1974, the total potentially hazardous waste generated by domestic underground mines was about 197,000 MT (217,000 tons) of waste rock.

Concentrating Operation Wastes. In the uranium ore concentrators, the uranium values are recovered from the crude ore and concentrated to yield an intermediate, semirefined product containing U_3O_8 or $Na_2U_2O_7$.19/ This product, which is commonly referred to in the industry as yellowcake, is shipped to refineries and reprocessed to obtain either uranium metal, UO_2, or UF_6 for use in the nuclear industry. This study is concerned only with the processes and wastes involved in mining and concentrating the ore to produce yellowcake.

Basic Technology of the Ore Concentrating Industry. The basic steps required in all uranium ore processing consist of: (1) ore preparation; (2) ore concentration; (3) yellowcake recovery.16/ Ore preparation includes crushing and grinding and possibly drying or roasting to improve handling or solubility properties. Concentration is carried out by hydrometallurgical leaching methods, using either dilute sulfuric acid or alkaline carbonate solutions.16/ The uranium concentrate is recovered from solution by chemical precipitation, dewatered, ground, and packaged for shipment.

The uranium ore dressing industry can be divided into two basic types of concentrating operations which are characterized by the leaching process used.

- Concentrators which use acid leaching or combined acid-and-alkaline leaching operations.

- Concentrators which use alkaline leaching only.

Acid leaching is used for ores containing less than 12 percent lime, and alkaline leaching is especially useful for the high-lime ores that would consume excessive amounts of acid.20/

Concentration and purification of the uranium values from dilute and impure leach solutions are required for production of a final high-grade, uranium concentrate product.21/ The two principal techniques utilized for concentration and purification are solvent extraction and resin ion exchange, used either individually or in a two-stage series. Solvent extraction is used to treat only clarified acid solutions, while the ion exchange process is applicable to the treatment of both ore pulps and clarified solutions in either acid or alkaline solution. Alkaline leaching operations are frequently sufficiently selective to permit elimination of the purification step.22/

The uranium ore concentrating industry is highly diversified. The processes used vary among the domestic plants due largely to differences in chemical composition of the ore, the cost and availability of leaching chemicals, and the presence of other metal constituents to be recovered as by-products. A tabulation of the producing companies and an identification of the concentrator operations used bv the active domestic ore concentrator plants during 1974, are shown in Table 81.

The data in Table 81 show that the daily feed ore processing capacity of the concentrator plants ranged from about 408 MT (450 tons) to 6,350 MT (7,000 tons) in 1974. There were 16 active concentrator plants in 1974. The ores processed commercially in the United States generally contain 0.1 to 0.5 percent U_3O_8; the average concentration is about 0.2 percent.[22/]

An analysis of the data in Table 81 indicates that the percentage distribution of types of concentrator plants active in 1974 was that shown in Table 82.

The data presented in Table 82 show that 12 of the 16 plants used an acid leach process and represented 79.6 percent of total concentrator capacity. Two concentrator plants utilized an alkaline leach process and accounted for 13.7 percent of production capacity, while another two plants used a combination acid-and-alkaline leach process and contributed 6.7 percent of the total concentrator capacity.

The acid leach facilities active in 1974 can be further subdivided, as shown in Table 82, into four plants using solvent extraction (41.8 percent of total concentrator capacity), three plants using solvent extraction plus ion exchange (11.5 percent of total concentrator capacity), and five plants using ion exchange (26.3 percent of total concentrator capacity). Thus, the entire industry is represented by either acid leach or alkaline leach processes with a number of variations in the concentrating and purification steps in the acid leach processes. The acid leach process dominated the industry in 1974 and represented about 80 percent of the total concentrator capacity.

The 16 active concentrator plants in 1974 were sited in six western states. Three plants were located in New Mexico, seven plants in Wyoming, two in Colorado, two in Utah, and single plants in Texas and Washington. The combined operations in the States of Wyoming and New Mexico accounted for 77.1 percent of total concentrator capacity (20,460 MT of ore per day).

The data base for the uranium ore mining and concentrating industry collected by Midwest Research Institute (MRI), by site visits and written questionnaires, did not represent a sufficiently wide cross-section to make it truly representative of the industry category. Therefore, in preparing a characterization of representative concentrator processes and their waste generation, it was necessary to use technical data available in the current published literature concerning model plants for processing uranium ores.[19/]

TABLE 81

ACTIVE U.S. URANIUM ORE CONCENTRATOR PLANTS IN 1974

					Operations used§										
Company†/ concentrator location†/ EPA region	Date started‡	Nominal capacity metric tons of ore/day†	Type of mine‡ Under-ground	Open-pit	Average ore grade % U_3O_8‡	Leach	Liquid-solid separa-tion	Ion exchange	Solvent extrac-tion	Eluex†† process	Precipi-tation	Remarks	Yellow cake, % U_3O_8	Overall uranium recovery %	By-product vanadium recovery
Anaconda Co./New Mexico/VI	1956	2,722		X	0.34	Acid	CCD	RIP			MgO		87		
United Nuclear-Homestake Partners/ New Mexico/VI	1958	3,175	X		0.21	Alkali	DF				NaOH, H_2SO_4, NH_3			95	Yes
Kerr-McGee Corporation/New Mexico/VI	1958	6,350	X			Acid	CCD		Amine		NH_3			> 97	
Conoco-Pioneer Nuclear/Texas/VI	1974	1,588		X		Acid			Amine		NH_3		97		
Cotter Corp./Colorado/VIII	1958	408	X			Acid	DF		**				80	93-96	
						Alkaline	DF				NaOH, H_2SO_4, NH_3	Dual process			
Union Carbide Corp./Colorado/VIII	1976	1,814	X		0.15	Acid	CCD	IX			NH_3		90	93-96	Yes
Atlas Corp./Utah/VIII	1956	1,361	X			Acid	CCD		Amine††						
						Alkaline		RIP			NH_3	Dual process			
Rio Algom/Utah/VIII	1972	454	X			Alkaline	DF				NaOH, H_2SO_4, H_2O_2				
Union Carbide Corp./Wyoming/VIII	1960	907		X		Acid		RIP			NH_3			> 90	
Utah International, Inc./Wyoming/VIII Gas Hills	1958	1,089		X	0.20	Acid	CCD	IX	Amine	Yes	NH_3		90	95	
Utah International, Inc./Wyoming/VIII Shirley Basin	1972	1,089		X		Acid	CCD	IX			NH_3 or MgO				
Petrotomics Co./Wyoming/VIII	1962	1,361		X		Acid	CCD		Amine		NH_3		97	97	
Western Nuclear, Inc./Wyoming/VIII	1957	1,089	X	X		Acid	CCD	RIP	Amine	Yes	NH_3				
Exxon, USA/Wyoming/VIII	1972	1,814		X		Acid	CCD		Amine		NH_3		95		
Federal American Partners/Wyoming/VIII	1959	862		X		Acid		RIP	Amine	Yes	NH_3				
Dawn Mining Co./Washington/X	1968	454		X	0.298	Acid	CCD	IX							
		26,537													

† Data taken from References 6 and 19.

‡ Data taken from References 19 and 20.

§ Data taken from data sources 19 and 20. The nomenclature for abbreviations listed under table headings are: CCD - Countercurrent Decantation RIP - Resin in ulp

DF - Drum Filters IX - Column Ion Exchange

** In the dual process, utilizing both acid and alkaline leach steps, solvent extraction is conducted using an alkyl phosphoric acid extractant, di(2-ethylhexyl) phosphoric acid. The loaded strip solution is combined with the filtered alkaline leach solution from the alkaline concentrator process for precipitation.

†† In this dual process, the loaded strip solution is combined with the loaded strip solution from the alkaline RIP circuit for precipitation.

‡‡ A combination ion exchange--solvent extraction process (sulfuric acid elution of resin followed by solvent extraction).

TABLE 82

DISTRIBUTION OF URANIUM ORE CONCENTRATOR PLANTS OPERATED IN 1974 BY TYPE OF PROCESS AND PERCENT OF TOTAL FEED ORE CAPACITY*

	Number	% of Total concentrator capacity
Plants using acid leach process	12	79.6
Plants using acid leach and alkaline leach processes (dual process)	2	6.7
Plants using alkaline leach process (with no purification step)	2	13.7
Total	16	100.0
Acid leach plants:		
Plants using solvent extraction	4	41.8
Plants using solvent extraction plus ion exchange	3	11.5
Plants using ion exchange	5	26.3
Total	12	79.6

* Compiled from data in Table 81.

Descriptions of Representative Processes. Detailed descriptions of the acid leach process and the alkaline leach process for concentrating uranium ore are presented in the following subsections.

Acid Leach Process. A model acid leach concentrator plant described in the technical literature was selected as being representative of this type of process.[19] An analysis of 1974 data (Tables 81 and 82) shows that there are two major subcategories for this process: acid leach solvent extraction, representing about 53 percent of total acid leach concentrator capacity, and acid leach-ion exchange accounting for about 33 percent of total plant capacity. A third subcategory is a dual process combining acid and alkaline leach systems. The two major categories taken together, represent most (86 percent) of the acid leach processes.

Description of Representative Process. A detailed discussion of a model plant reported in the technical literature[19] for the acid leach solvent extraction process is given in this section. The use of this type of concentrator plant appears to be the trend of the future since four out of five of the new acid leach plants constructed in the western world in the last 5 years were of this type.[19] Also, many of the processing variables in acid leach plants affecting the radiological waste are the same for both solvent extraction and ion exchange systems.[19] The common processing items are:

* The system for ore size reduction and chemical leaching.

* The quantity and composition of the solid wastes generated.

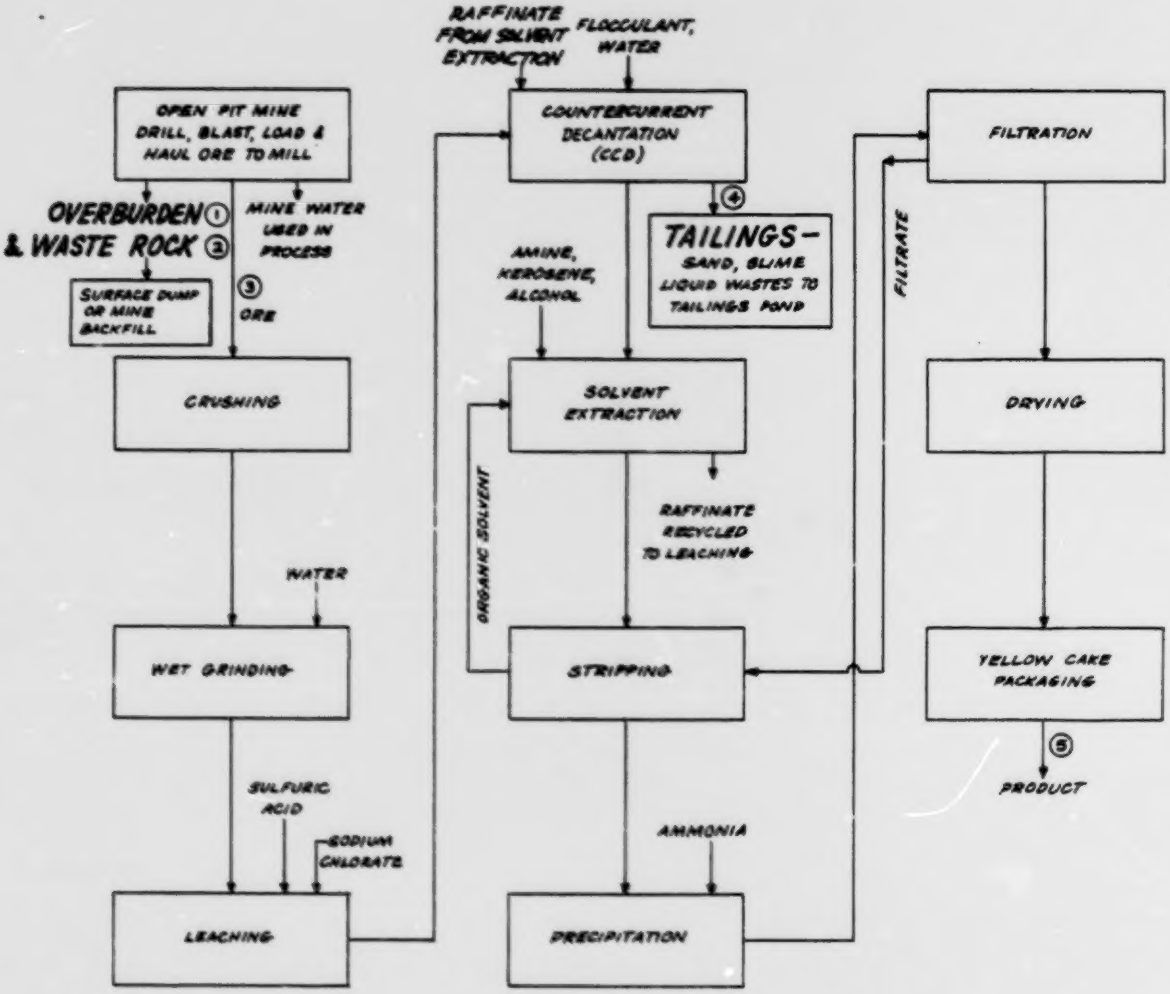
* The quantity of radionuclides in the liquid effluent wastes.

* The treatment and disposal methods used for solid, liquid, and gaseous wastes.

* The quantity and composition of airborne effluents.

Therefore, it is considered that the acid leach solvent extraction process also basically represents the operations and waste generation for acid-leach with ion exchange.

A representative flow diagram for the acid leach solvent extraction process is shown in Figure 24. Uranium ore from either an open-pit mine or an underground mine can be used in this process. An open-pit mine is shown as an example in this study.


RAFFINATE FROM SOLVENT EXTRACTION
FLOCCULANT, WATER
OPEN PIT MINE DRILL, BLAST, LOAD & HAUL ORE TO MILL
OVERBURDEN ①
& WASTE ROCK ②
MINE WATER USED IN PROCESS
SURFACE DUMP OR MINE BACKFILL
③
ORE
CRUSHING
WATER
WET GRINDING
SULFURIC ACID
SODIUM CHLORATE
LEACHING
COUNTERCURRENT DECANTATION (CCD)
④
TAILINGS—
SAND, SLIME LIQUID WASTES TO TAILINGS POND
AMINE, KEROSENE, ALCOHOL
SOLVENT EXTRACTION
ORGANIC SOLVENT
RAFFINATE RECYCLED TO LEACHING
STRIPPING
AMMONIA
PRECIPITATION
FILTRATE
FILTRATION
DRYING
YELLOW CAKE PACKAGING
⑤
PRODUCT

MATERIALS BALANCE AND COMPOSITION

DATA ITEMS	① OVER-BURDEN	② WASTE ROCK	③ CRUDE ORE (DRY)	④ TAILINGS DRY SOLIDS	④ TAILINGS LIQUID	⑤ YELLOW CAKE PRODUCT
Annual Quantity, Metric TPY	19,418,000	425,700	662,200	662,200	993,300	1,339
Partial Analysis:						
U_3O_8, Wt.,%			0.20	0.018		90
Radium, μCi/g U_3O_8						5.5×10^{-4}
Thorium, μCi/g U_3O_8						1.4×10^{-2}
U_{nat}, pCi/g				52		
^{226}Ra, pCi/g				567		
^{230}Th, pCi/g				567		
^{234}Th, pCi/g				52		
U_{nat}, pCi/liter					6,700	
^{226}Ra, pCi/liter					500	
^{230}Th, pCi/liter					190,000	
Calcium (Ca) g/liter					0.5	
Iron (Fe) g/liter					1.0	
Aluminum (Al) g/liter					2.0	
Ammonia (NH_3) g/liter					0.5	
Sodium (Na) g/liter					0.2	
Arsenic (As) g/liter					0.0002	
Fluoride (F) g/liter					0.005	
Vanadium (V) g/liter					0.0001	
Sulfate (SO_4^{2-}) g/liter					30	
Chloride (Cl^-) g/liter					0.3	
Total Dissolved Solids, g/liter					35.0	
pH					2	

Source: Reference 36.

PROCESS ADDITIVES

NAME	LB/TON OF ORE PROCESSED	G/MT OF ORE PROCESSED
Sulfuric Acid	73	36,500
Sodium Chlorate	2.2	1,100
Ammonia	2.3	1,150
Amine*	0.035	17.5
Alcohol†	0.06	30.0
Kerosene	0.64	320.0
Flocculant	0.16	80.0
Total	78.40	39,198

* Alamine 336 and Adogen 364 are used interchangeably.
† Isodecanol and Tridecanol are used interchangeably.

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Figure 24. Mining and concentrating of uranium ore--acid leach process.

The model concentrator was selected to have a daily capacity of 1,814 MT (2,000 tons) of feed ore containing 0.20 percent U_3O_8. The concentrator is assumed to operate continuously for 364 days/year.[19] A discussion of the process used in the model concentrator follows.

Ore delivered by dump trucks is passed through a bar screen (grizzly) to a primary crusher which reduces the ore to -1.25 cm (1/2 in.) size. Oversize material is recycled to the crusher. The crushed ore is elevated and moved on a rubber belt conveyor to storage bins which are vented through a dust collection unit (orifice dust collector) to the atmosphere. Air exhaust hoods are located on the crusher, at the screens, and at each ore transfer point. The air is passed through a dust separator unit (orifice dust collector) and then discharged to the atmosphere through a roof vent.[19] Collected dust is sent to processing; detailed data on dust collection equipment and dust quantities are given in Reference 19.

Crushed ore, drawn from storage bins, is wet-ground in ball mills as mixture containing about 65 percent solids (by weight).[19] Ore ground to pass a 28-mesh screen is discharged into a chemical leaching system consisting of several mechanically agitated tanks in series with a total residence time of about 7 hr. In some other plants, air agitation is employed in tall vessels (called Pachucas) as a substitute leaching system. Sulfuric acid and an oxidant, sodium chlorate, are added continuously to the slurry discharged from the ball mills. To obtain the desired high leaching efficiencies, it is necessary to oxidize any tetravalent uranium present in the ore to the hexavalent state. Sodium chlorate ($NaClO_3$) is a typical oxidant. Manganese dioxide is also a suitable oxidizer. A typical leach reaction is:

$$3\ H_2SO_4 + NaClO_3 + 3\ UO_2 \longrightarrow 3\ UO_2SO_4 + NaCl + 3\ H_2O$$

The ore is leached continuously at atmospheric pressure and at slightly above room temperature. The pH of the leach slurry is usually maintained between 0.5 and 2.0.[20] In this step, most of the uranium values in the ground ore are dissolved along with other elements which may be contained in the ore such as some radionuclides, iron, aluminum, and many other impurities. The ore is ground prior to leaching to permit a much faster and more efficient extraction of uranium because of the greatly increased surface area in the ground ore particles.

Following the leaching step, the leach liquor (containing undissolved suspended solids) is sent to a countercurrent decantation (CCD) thickener operation.[19] Flocculant is added to the thickener to promote settling of solids. The CCD system serves to effect a separation of the uranium-bearing liquid from the undissolved solids. Water is added to wash uranium solution from undissolved solids. Underflow (slurry) from the CCD system is passed through liquid cyclone separators (hydroclones) to separate the coarse sand fraction, and the sand is washed in a series of classifiers. The overflow from the classifiers is combined with the hydroclone overflow, and the resulting materials (slimes) are washed with fresh water and recycled raffinate in a series of thickeners using flocculant to promote settling. The washed slimes and sands are pumped as a tailings slurry to a tailings pond. The weight ratio of dry solids in the tailings slurry to ore is 1 to 1; the composition of the dry solids is 70 percent sands and 30 percent slimes. The total weight of waste solution accompanying the sands and slimes to the tailings pond is 1.5 times the weight of the ore processed.[19]

Leach solution discharged by the countercurrent decantation system is sent to a solvent extraction (SX) step. Uranium is recovered from the leach liquor by countercurrent contact in several extraction stages in series (e.g., four stages) with a long-chain amine dissolved in kerosene.[19]

This process is based on the property of the organic solvent, which is immiscible in water, to form a complex with the uranium. This complex is soluble in an excess of the solvent.[19] When aqueous leach solution containing solubilized uranium compounds is contacted with the organic solvent, the uranium is distributed between the aqueous and organic phases. Under properly controlled process conditions, the uranium is extracted almost quantitatively into the organic phase, while most of the other constituents (impurities) of the leach liquor remain in the aqueous phase. After mixing to provide adequate contact, the mixture of solutions is removed to a settling tank, where the lighter, uranium-loaded layer rises to the surface and is separated by decantation. These operations are conducted on a continuous basis in a series of mixer and settler tanks. Aqueous raffinate from the solvent extraction operation is recycled to the countercurrent decantation step.[19]

The uranium loaded organic solvent discharged from the solvent extraction is treated in a stripping operation to separate the uranium from the organic solvent.[19] The uranium is stripped from the solvent in several stages (e.g., four stages) with an aqueous solution of ammonium sulfate. The recovered organic solvent is recycled back to the solvent extraction step.[19]

Uranium loaded aqueous solution from the stripping operation is treated by chemical precipitation to separate uranium values from the liquor.[19] The uranium is precipitated by addition of gaseous ammonia to the strip solution, concentrated, and partially washed in thickeners.[19]

The uranium-bearing aqueous slurry discharged from the precipitation step is collected and washed on filters.[19] The wet uranium-bearing solids discharged from the filters are dried in a continuous steam-heated dryer. The dried uranium precipitate (yellowcake) is packaged in 55-gal steel drums for shipment to a refinery. The yellowcake product contains about 90 percent U_3O_8, and the overall recovery of uranium as product is 91 percent of that contained in the feed ore.[19] The literature indicates that the thorium content of the yellowcake is about 1.4×10^{-2} µCi/g U_3O_8 (i.e., 5 percent of the total thorium in the ore), and that the radium content is 5.5×10^{-4} µCi/g U_3O_8 (0.2 percent of total radium in the ore).[19] No other significant radionuclides are contained in the product.

The air streams from the U_3O_8 precipitate dryer and the hoods over the packaging area are combined and passed through a dust collector (wet impingement dust collector).[19] Collected dust is recycled to the process.

Any liquid spillage or leakage throughout the concentrator plant is collected in floor sumps and then returned to the appropriate process step. The only liquid waste stream is that leaving with the sands and slimes to the tailings area.[19]

Mass Balances of Materials. Data for a mass balance of materials involved in the acid leach concentrating process are shown in Figure 24. These data apply for a model plant (not an existing facility) reported in the current technical literature.[19] This model is considered to be representative of all domestic acid leach processes.[19]

The data for this model show that on an annual basis, the consumption of 662,200 MT (729,950 tons) of feed ore corresponds to an identical quantity of tailing solids, 1,339 MT (1,476 tons) of product, and 993,300 MT (1,094,900 tons) of waste tailing liquid.

Process additives used include sulfuric acid, sodium chlorate, ammonia, amine, alcohol, kerosene, and flocculant. The amine, alcohol, and kerosene are recovered and reused in the process, while the remaining materials and their reaction products are distributed among the waste materials. The consumption of additives amounts to 39,200 g/MT of feed ore (78.4 lb/ton).

The composition of the dry waste solids is 30 percent slimes and 70 percent sands (by weight).[19] The slimes fraction contains 85 percent of the insoluble radioactive materials. The uranium distribution (originating with the feed ore) is 91 percent in the yellowcake product, 7.6 percent in the slimes, and 1.4 percent in the sands.[19] Fifty percent of the thorium dissolves during concentrating and ultimately crystallizes in the slimes fraction, while 50 percent is insoluble in the tailing.[19] Other radionuclides are insoluble.

Total production statistics for uranium-vanadium ores in 1974 are shown in Table 79. The estimated total quantity of concentrator dry tailings wastes were 5,715,000 MT (6,299,700 tons). Data in Table 82 show that 79.6 percent of the total domestic concentrator capacity was represented by concentrator plants using an acid leach process. The amount of solid waste generated is directly proportional to the quantity of feed ore consumed regardless of the type of concentrating process. Therefore, assuming an equal utilization of processing capacity by each concentrator plant, the quantity of land-disposed solid wastes from acid leach processes can be estimated as 79.6 percent of 5,715,000 or 4,549,000 MT (5,014,400 tons) in 1974. Based on wastewater data from written questionnaires and estimates by MRI, the total wastewater discharged to tailings ponds (including some direct discharge of mine wastewaters) in 1974 for all acid leach processes was 10,201,000 MT (11,245,000 tons).

The other land-disposed wastes for the industry in 1974 were attributable to the operation of dual concentrator processes using both an acid leach and an alkaline leach. In 1974, there were two active dual processes which accounted for 6.7 percent of the total feed ore consumed. The estimated quantity of solid wastes from dual processes during that year is: 6.7 percent of 5,715,000 or 382,900 MT (422,100 tons); the estimated amount of liquid wastes accompanying the solid wastes to land disposal is 688,000 MT (758,000 tons).

Description of Individual Waste Streams. The only concentrator waste which is land-disposed is the tailings slurry containing sands, slimes, and liquid. Airborne effluents containing ore dust, radon gas or yellowcake dust are discharged to the atmosphere from various operations in the concentrator plant, such as ore crushing and screening and product drying.19/

Data on the composition of waste generated in a model acid leach-solvent extraction plant are presented in Figure 24. In this model plant, the dry solids fraction of the tailings contains 0.018 percent of U_3O_8, 52 pCi/g of natural uranium, 567 pCi/g each of radium-226 and of thorium-230, and 52 pCi/g of thorium-234.

As shown in Figure 24, the liquid fraction of the tailings slurry (for the model plant) contains various metals dissolved from the uranium ore along with chemical additives (or reaction products) introduced in the concentrator process.

Identification of Potentially Hazardous Waste Materials. The entire concentrator tailings, which are land-disposed, are potentially hazardous because they contain radioactive materials in concentrations above the background level.

Alkaline Leach Process. A model alkaline concentrator plant reported in the technical literature was selected as representative of this type process.19/

Description of Representative Process. A representative flow for the alkaline leach process is shown in Figure 25. Uranium ore from either an open-pit mine or an underground mine can be used with this process. An underground mine is shown as an example in the study.

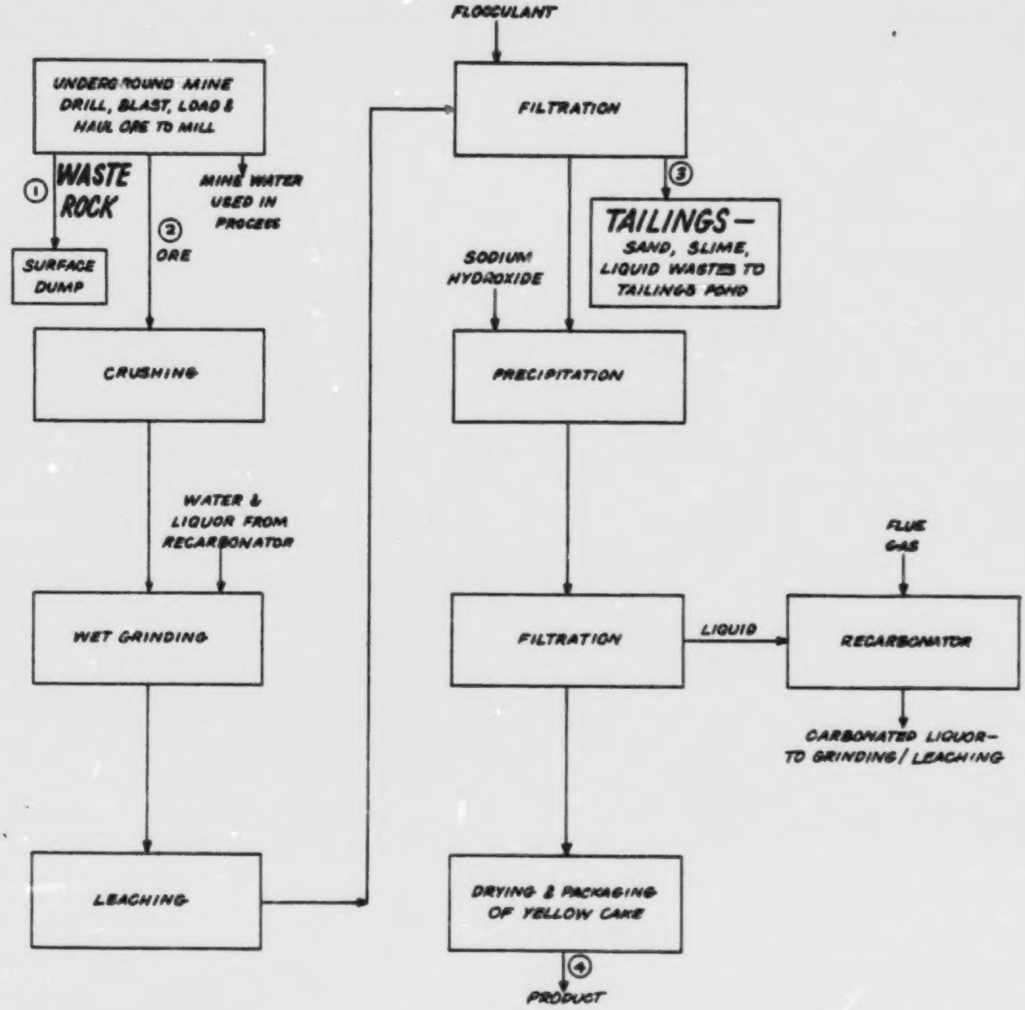
The model concentrator plant has a daily capacity of 1,814 MT (2,000 tons) of feed ore containing 0.20 percent U_3O_8.19/ The concentrator is assumed to operate continuously for 364 days/year.19/ A discussion of the process used in the model concentrator follows.

The operations involved for ore receiving, conveying crushing and ore storage facilities are the same as those described previously for the acid leach mill including dust separation from air streams.19/ Therefore, the description of these activities is not repeated here.

The crushed ore is ground to provide a finely divided ore suitable for efficient leaching in subsequent processing. The wet-grounding system consists of a ball mill operated in closed circuit with a classifer.19/ The grinding is carried out at a solids content of 65 percent (by weight) in a sodium carbonate-bicarbonate solution. The ore discharged from the system is finely ground and the average particle size is about 95 percent -48 mesh, and 35 percent -200 mesh.

The ground ore is leached in a two-stage system consisting of a 5-hr leach at a pressure of 65 psig (in autoclaves) and 200 F, followed by an 18-hr leach at atmospheric pressure and 185 F.19/ The alkaline leaching system requires the oxidation of any tetravalent uranium contained in the ore to the hexavalent state using oxygen available in air supplied to the leaching vessels. Hexavalent uranium dissolves in the presence of carbonate alkalinity to form a uranyl tricarbonate complex ion. Leaching is carried out in an aerated solution containing sodium carbonate and sodium bicarbonate.19/

Undissolved solids are separated and washed free of uranium by three stages in series of countercurrent filtration (i.e., a sequence of filtering and repulping operations). The solids (filter cake), which are separated from the final filtration step, consist solely of equal parts (by weight) of sands and slimes; these solid wastes are repulped with fresh water and pumped to a tailings pond. The weight ratio of solid wastes to ore is 1 to 1. The weight of waste solution sent to the tailings ponds is 1.05 times the weight of the feed ore processed.19/


FLOCCULANT
UNDERGROUND MINE DRILL, BLAST, LOAD & HAUL ORE TO MILL
①
WASTE ROCK
SURFACE DUMP
②
ORE
MINE WATER USED IN PROCESS
CRUSHING
WATER & LIQUOR FROM RECARBONATOR
WET GRINDING
LEACHING
FILTRATION
③
TAILINGS — SAND, SLIME, LIQUID WASTES TO TAILINGS POND
SODIUM HYDROXIDE
PRECIPITATION
FLUE GAS
FILTRATION
LIQUID
RECARBONATOR
CARBONATED LIQUOR— TO GRINDING/LEACHING
DRYING & PACKAGING OF YELLOW CAKE
④
PRODUCT

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS	① WASTE ROCK	② CRUDE ORE (DRY)	③ TAILINGS DRY SOLIDS	③ TAILINGS LIQUID	④ YELLOW CAKE PRODUCT
Annual Quantity, Metric TPY	51.000	662.200	662.200	695.300	1.449
Partial Analysis:					
U_3O_8, Wt %		0.20	0.014		85
Radium, μCi/g U_3O_8					5.5×10^{-3}
Unat. pCi/g			40		
226 Ra. pCi/g			560		
230 Th. pCi/g			565		
234 Th. pCi/g			40		
Unat. pCi/liter				10.000	
226 Ra. pCi/liter				100	
230 Th. pCi/liter				20	
Iron (Fe) g/liter				0.0005	
Aluminum (Al) g/liter				1.0	
Sodium (Na) g/liter				3.0	
Arsenic (As) g/liter				0.0002	
Fluoride (F) g/liter				0.002	
Vanadium (V) g/liter				0.0001	
Sulfate (SO_4^{2-}) g/liter				2.0	
Chloride (Cl^-) g/liter				1.0	
Carbonate (CO_3^{2-}) g/liter				6.0	
Total Dissolved Solids, g/liter				12.0	
pH				10	

Source: Reference 36.

PROCESS ADDITIVES

NAME	LB/TON OF ORE PROCESSED	G/MT OF ORE PROCESSED
Sodium Hydroxide	15	7.500
Sodium Carbonate	12	6.000
Total	27	13.000

Figure 25. Mining and concentrating of uranium ore--alkaline leach process.

Uranium is recovered from the leach solution by addition of sodium hydroxide, which forms insoluble sodium diuranate.[19] In the clarified solution, the uranium is present as a complex uranyl tricarbonate ion which is stable at a pH less than 11. To accomplish precipitation of the uranium values, the pH of the solution is raised to above 12 by the addition of sodium hydroxide. At this pH level, the complex ion decomposes to form sodium diuranate and sodium carbonate. The decomposition reaction is:

$$2\ Na_4UO_2(CO_3)_3 + 6\ NaOH \longrightarrow Na_2U_2O_7 + 6\ Na_2CO_3 + 3\ H_2O$$

The precipitation is carried out at atmospheric pressure and 180 F in a series of agitated tanks.[19]

The caustic barren solution from the precipitation circuit contains sodium carbonate and a small amount of sodium hydroxide. To permit reuse of this barren solution, the caustic must be converted to sodium carbonate and sodium bicarbonate. This required conversion is carried out in packed towers (recarbonators) in which caustic barren solution is contacted with boiler flue gas. The carbon dioxide in the flue gas neutralizes the sodium hydroxide and converts some of the carbonate to bicarbonate. This recarbonated barren solution is sent to the filtration circuit for use as a wash liquor.[19]

The precipitated sodium diuranate (yellowcake) is filtered, washed, and dried in a steam-heated dryer.[19] The dried product is packaged in 55-gal. drums for shipment. Each drum contains a maximum of 455 kg (1,000 lb).

Off-gas from the drying and packaging area is passed through a dust separation system (wet impingement dust collector) before discharge to a roof stack. The overall recovery of uranium is 93 percent of that contained in the feed ore, and the U_3O_8 content in the product is 85 percent.[19] The estimated radium content in the yellowcake is 5.5×10^{-3} µCi/g U_3O_8, which represents about 1.8 percent of that in the feed ore.[19] No other significant radionuclides are present in the yellowcake.

For alkaline leach processes which involve the recovery of by-product vanadium values, some minor process modifications are required in order to maximize the production of vanadium values as well as yellowcake. For example, removal of the vanadium and the carbonate from the initial yellowcake product is accomplished by a roasting treatment at 1600 F followed by water leaching. The water leaching serves to dissolve the carbonate and vanadium in the yellowcake. The vanadium is usually recovered as a solution containing V_2O_5 values and sold to vanadium producers.

In order to meet specifications set by some purchasers for sodium content in yellowcake, an additional process is required. After removal of vanadium, yellowcake containing about 7.5 percent sodium is dissolved with sulfuric acid at a pH of 2.2. Ammonia is then added to this leach solution to maintain a pH of 7.4 and precipitate yellowcake containing less than 0.5 percent sodium. This product is then filtered, dried, and packaged.

Mass Balance of Materials. Data for a mass balance of materials involved in the alkaline leach concentrating process are shown in Figure 25. These data apply for a model plant described in the technical literature.19/ It should be noted that this model concentrator does not represent the design for any particular existing facility; however, the model is considered to be representative of all domestic alkaline leach processes.19/

The data for this model show that on an annual basis, the consumption of 662,200 MT (729,950 tons) of feed ore corresponds to 1,449 MT (1,597 tons) of product, 662,200 MT (729,950 tons) of tailing solids and 695,300 MT (766,440 tons) of tailing wastewater.

Process additives include sodium hydroxide and sodium carbonate. The total consumption rate of additives amounts to 13,500 g/MT of ore processed (27 lb/ton ore).19/

The dry waste solids for the model plant are composed of equal parts by weight of slimes and sands. The slimes fraction contains 85 percent of the insoluble radioactive materials.19/ The recovery of uranium in product (yellowcake containing 85 percent U_3O_8) is 93 percent of that in the feed ore; 1 percent remains in the sands and 6 percent in the slimes.19/ The yellowcake product contains 2 percent of the radium originally in the feed ore, and 98 percent of the radium is present as an insoluble form in the tailings. The other radionuclides remain insoluble during leaching and leave the concentrator with the tailings solids.19/

Total production statistics for uranium-vanadium ores in 1974 are shown in Table 79, p. 170. From data in Table 82 showing that 13.7 percent of the concentrator capacity was from alkaline leach plants, it is possible to estimate the quantity of land-disposed solid wastes from alkaline leach processes in 1974 as 13.7 percent of 5,715,000 or 783,000 MT (863,100 tons). The quantity of total wastewater generated by alkaline leach processes during 1974 is estimated to be 777,000 MT (857,000 tons).

Description of Individual Waste Streams. The only concentrator waste which is land-disposed is the tailings slurry containing sands, slimes, and liquid. Airborne effluents containing ore dust, radon gas or yellowcake dust are discharged to the atmosphere.

Data on the composition of waste generated in a model alkaline leach concentrator plant are shown in Figure 25. For the model plant, the dry solids fraction of the tailings contains 0.014 percent U_3O_8, 40 pCi/g of natural uranium, 560 pCi/g of radium-226, 565 pCi/g of thorium-230, and 40 pCi/g of thorium-234.

As shown in Figure 25, the liquid portion of the tailings slurry contains natural uranium and radionuclides (radium-226 and thorium). This liquid also contains a number of metals including iron, aluminum, sodium, arsenic, and vanadium as well as fluoride, sulfate and chloride compounds. The total dissolved solids content in the liquid is about 12 g/liter and the liquor has an average pH of about 12.

Identification of Potentially Hazardous Waste Streams. The entire concentrator tailings, which are a land-disposed waste, are potentially hazardous because they contain radioactive materials in concentrations above the background level.

Table 83 shows the quantities of potentially hazardous wastes generated by mining and concentrating uranium-vanadium ores in 1974. The national total of potentially hazardous dry waste was 7,938,000 MT (8,750,000 tons) consisting of 2,223,000 MT (2,450,000 tons) of waste rock and 5,715,000 MT (6,300,000 tons) of concentrator tailings solids. The total potentially hazardous dry waste accounted for 7.8 percent of the total dry waste. In addition, the estimated quantity of potentially hazardous wastewater discharged for disposal in tailings ponds was 11,667,000 MT (12,861,000 tons). Therefore, the total quantity of potentially hazardous wastes generated by this industry in 1974 was 19,605,000 MT (21,611,000 tons).

Modifications of Concentrator Processes. Several modified recovery methods not discussed in the previous process descriptions can be employed, following the leaching operations, to yield a uranium concentrate. These modifications are:

* Ion exchange circuits.

* Alternate solvent extraction systems (Dapex, Amex, Eluex processes).

* Alternate precipitation methods, using sodium hydroxide, gaseous ammonia, hydrogen peroxide, or magnesia.

* Secondary metal recovery.

The waste products, and potentially hazardous wastes, from these modifications have been treated in previous subsections which present information as the basic acid and alkaline leach processes, and therefore will not be discussed in this subsection.

TABLE 83

TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING URANIUM-VANADIUM ORES FOR SIC 1094 IN 1974
(METRIC)

		Dry waste weight						Dry weight		Wet weight	
				Waste rock		Overburden		Concentrator tailings		Concentrator tailings	
State	EPA Region	Total process waste (10^3 TPY)	Total potentially hazardous waste generated (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)
New Mexico		15,748	3,576	1,127	1,127	12,172	0	2,449	2,449	6,930	6,930
Texas		8,114	851	484	484	7,263	0	367	367	2,276	2,276
Total	VI	23,862	4,427	1,611	1,611	19,435	0	2,816	2,816	9,206	9,206
Colorado		514	514	0	0	0	0	514	514	1,092	1,092
Utah		790	458	38	38	332	0	420	420	932	932
Wyoming		74,437	2,213	353	353	72,224	0	1,860	1,860	5,889	5,889
Total	VIII	75,741	3,185	391	391	72,556	0	2,794	2,794	7,913	7,913
Washington	X	1,970	326	221	221	1,644	0	105	105	263	263
National		101,573	7,938	2,223	2,223	93,635	0	5,715	5,715	17,382	17,382

Source: Reference 36.

Ion Exchange Systems. These systems are used to concentrate and purify the uranium leaching solution. Strong and intermediate base anion-type resins are loaded from either a sulfuric acid or a carbonate leach feed solution. The loaded resin is stripped with a chloride, nitrate, bicarbonate or an ammonium sulfate-sulfuric acid solution.[17]

Four types of ion exchange circuits are used by uranium mills. They are: (1) fixed bed, (2) moving bed, (3) continuous resin-in-pulp (RIP) process, (4) basket resin-in-pulp process.[17] The fixed-bed type consists of stationary columns packed with resin. As the feed solution passes through the columns, uranium is absorbed on the resin. The resin is washed and the uranium desorbed. The moving bed column circuit has stationary columns, but the resin is transferred to different columns to perform loading, washing, and eluting operations. In the continuous resin-in-pulp (RIP) process, sorption, washing, and desorption are conducted by contacting the resin and process solutions in a series of tanks. The resin and solution flow countercurrently in the tanks and are separated by screens. Forced air is used for agitation. The basket resin-in-pulp process employs a series of resin-filled cubical baskets that are jigged up and down in the process solution contained in tanks. Feed slurry, eluant, and wash solution are passed through the tanks to extract the uranium. The RIP process represents an exception to the separation procedure in that suspended solids do not have to be removed from the leach slurry.[17]

In 1974, there were four concentrator plants which used column ion exchange units. Taken together, these process units were representative of about 17 percent of the total domestic concentrator feed ore capacity and the same percentage of the total solid wastes generated by concentrators and land-disposed.

During 1974, five concentrator plants used the RIP process. These five plants represented about 26 percent of the total concentrator feed ore capacity and the same percentage of the total solid waste which was land-disposed by concentrators.

Alternate Solvent Extraction Systems. Two solvent extraction processes, the Dapex and the Amex, are currently used to concentrate uranium in about one-half of the concentrators.[17] The processes are efficient but require that the leach solution be essentially free of solids.

The uranium is extracted from clarified feed solution into an organic solvent, followed by stripping operations to yield a uranium-bearing aqueous phase.[17] The circuit consists of a series of tanks in which feed solution flows countercurrent to the organic solvent. Phase separation is improved by optimizing mixing rates of solutions and by installing settlers in the circuit.

The Dapex process uses the alkyl phosphoric acid extractant, di(2-ethylhexyl) phosphoric acid (EHPA) at a 4 percent concentration in kerosene with tributyl phosphate (TBP) added as a modifier to improve phase separation.[17] Long-chain alcohols, such as isodecanol, are also used as modifiers. The loaded organic phase is stripped of uranium with a sodium carbonate solution and recycled to the extraction stage of the circuit for reuse.

The Amex process consists of an amine extraction followed by stripping with ammonium sulfate, chloride or sodium carbonate solutions.[17] A 6 percent concentration of a tertiary amine, such as alamine-336, in a kerosene diluent is employed as the organic extractant. Isodecanol is added as a modifier. Stripping with ammonium sulfate at a pH of 4.0 to 4.3 eliminates sodium impurities.

The Eluex process is a combination ion exchange-solvent extraction process.[17] The eluate produced by sulfuric acid elution of the ion exchange resin is fed to either the Dapex or the Amex solvent extraction process. The Eluex process eliminates the need for nitrate and chloride reagents and yields a purer end product. In 1974, this process was used in three concentrator plants representing about 11.5 percent of total concentrator feed ore capacity and about 11.5 percent of the total concentrator solid wastes.

Alternate Precipitation Methods. Uranium is precipitated from solution by the addition of sodium hydroxide, gaseous ammonia, hydrogen peroxide or magnesia.[17] Several stages of precipitation at controlled pH are often used. In most concentrators (12 of 16, representing about 80 percent of the total concentrator feed ore capacity in 1974), gaseous ammonia is utilized as the precipitating agent.

Secondary Metal Recovery. In some instances, other metals are present in uranium ores in sufficient concentrations to permit economic extractions. For example, a mill in Utah extracts copper as a by-product by means of a flotation circuit. Vanadium is extracted at one mill in Colorado and one mill in New Mexico. Concentrations of molybdenum in ores mined in Texas may be sufficient to justify by-product recovery.

Potentially Hazardous Land-Disposed Waste in 1977 and 1983. Data from the U.S. Bureau of Mines Mineral Yearbook[23] show that for 1973 the total production of uranium concentrate was 12,007 MT (13,236 tons) of U_3O_8. Demand for U_3O_8 is expected to increase at an average annual rate of 18 percent through 1980.[24] Midwest Research Institute has estimated that this 18 percent annual increase will continue through 1983. It is not known whether future production levels will match the demand for uranium concentrate. However, on the basis of this estimated increased demand the domestic production of U_3O_8 could reach 23,280 MT (25,660 tons) of U_3O_8 in 1977 and 62,840 MT (69,270 tons) of U_3O_8 in 1983.

Based on these estimated production rates, calculations were made to provide projections of the total waste from domestic mining and concentrating of uranium-vanadium ores in 1977 and 1983. Previously developed data (Table 79) on mining and concentrating wastes for 1974 were used as a basis for these projections. It was assumed that no changes would occur in the regional distribution of mine and concentrator plants or the ratio of waste generated to ore mined and processed. Also, it was considered that the projected waste quantities would be directly proportional to the estimated U_3O_8 demand in a given year. Thus, factors based on relative U_3O_8 demand were applied to the 1974 waste data to obtain projections of waste quantities for 1977 and 1983.

Calculated data for total uranium-vanadium ore mining and concentrating wastes in 1977 are presented in Table 84. The estimated national potentially hazardous dry waste is 13,042,000 MT (14,300,000 tons) consisting of 3,652,000 MT (4,026,000 tons) of waste rock and 9,390,000 MT (10,351,000 tons) of concentrator tailing solids. Also, the estimated quantity of potentially hazardous wastewater in the concentrator tailings is 19,169,000 MT (21,130,000 tons). Thus, the total quantity of potentially hazardous wastes in 1977 is estimated to be 32,211,000 MT (35,507,000 tons).

Data concerning the uranium-vanadium ore mining and concentrating wastes in 1983 are shown in Table 85. The estimated total potentially hazardous dry waste is 35,207,000 MT (38,809,000 tons) consisting of 9,860,000 MT (10,869,000 tons) of waste rock and 25,347,000 MT (27,940,000 tons) of concentrator tailings solids. The estimated amount of potentially hazardous wastewater discharged to tailings ponds is 51,743,000 MT (57,037,000 tons). The total quantity of potentially hazardous wastes for 1983 is estimated to be 86,950,000 MT (95,846,000 tons).

Waste Characterization--Vanadium Ores.

Mining Wastes. The only operating vanadium mines in the United States are open-pit mines located near Hot Springs, Arkansas. At these open-pit mines, the vanadium-bearing component of the crude ore is vanadinite, $Pb_5(VO_4)_3Cl$, a reddish-brown material. Other minerals associated with the ore body are: illite, geothite, montroselite, pyroxene, quartz, potash feldspar, and kaolinite. This crude ore contains an average of about 1 percent V_2O_5.

TABLE 84

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING URANIUM-VANADIUM ORES FOR SIC 1094 IN 1977
(METRIC)

		Dry waste weight						Dry weight		Wet weight	
				Waste rock		Overburden		Concentrator tailings		Concentrator tailings	
State	EPA Region	Total process waste (10^3 TPY)	Total potentially hazardous waste generated (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)
New Mexico		25,874	5,875	1,852	1,852	19,999	0	4,023	4,023	11,385	11,385
Texas		13,331	1,398	795	795	11,933	0	603	603	3,739	3,739
Total	VI	39,205	7,273	2,647	2,647	31,932	0	4,626	4,626	15,124	15,124
Colorado		845	845	0	0	0	0	845	845	1,795	1,795
Utah		1,298	752	62	62	545	0	690	690	1,531	1,531
Wyoming		122,300	3,636	580	580	118,664	0	3,056	3,056	9,676	9,676
Total	VIII	124,443	5,233	642	642	119,209	0	4,591	4,591	13,002	13,002
Washington	X	3,237	536	363	363	2,701	0	173	173	433	433
National		166,885	13,042	3,652	3,652	153,842	0	9,390	9,390	28,559	28,559

Source: Reference 36.

TABLE 85

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING URANIUM-VANADIUM ORES FOR SIC 1094 IN 1983 (METRIC)

		Dry waste weight									
				Waste rock		Overburden		Dry weight		Wet weight	
								Concentrator tailings		Concentrator tailings	
State	EPA Region	Total process waste (10^3 TPY)	Total potentially hazardous waste generated (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)	Total (10^3 TPY)	Potentially hazardous (10^3 TPY)
New Mexico		69,842	15,859	4,998	4,998	53,983	0	10,861	10,861	30,734	30,734
Texas		35,986	3,775	2,147	2,147	32,211	0	1,628	1,628	10,094	10,094
Total	VI	105,828	19,634	7,145	7,145	86,194	0	12,489	12,489	40,828	40,828
Colorado		2,280	2,280	0	0	0	0	2,280	2,280	4,843	4,843
Utah		3,504	2,032	169	169	1,472	0	1,863	1,863	4,134	4,134
Wyoming		330,128	9,815	1,566	1,566	320,313	0	8,249	8,249	26,118	26,118
Total	VIII	335,912	14,127	1,735	1,735	321,785	0	12,392	12,392	35,095	35,095
Washington	X	8,737	1,446	980	980	7,291	0	466	466	1,167	1,167
National		450,477	35,207	9,860	9,860	415,270	0	25,347	25,347	77,090	77,090

Source: Reference 36.

Description of a Typical Process. At the Arkansas plant, three open-pit vanadium mines supply crude ore to a nearby mill. The total tonnage of ore mined annually is about 362,874 MT (400,000 tons) per year. The present operations involve the use of power shovels and scrapers, and the ore is moved to the mill by trucks. The overburden is drilled and blasted, and the solid waste is moved to nearby surface waste dumps, which are contoured and revegetated. The solid waste disposed of on land amounts to 907,185 MT (1,000,000 tons) per year. No mine backfilling is currently being done. A flow diagram for the mining process is shown in Figure 26.

Mass Balance of Materials. At this mine, an average of 2.38 MT (2.62 tons) of overburden and 0.125 MT (0.138 tons) of waste rock must be handled for every ton of crude ore produced. The production rate for crude ore is about 362,874 MT (400,000 tons) per year. Mass balance data are shown in Figure 27 and Table 86. Data on the ratio of wastes to ore and of wastes to product are presented in Table 87.

Description of Individual Waste Streams. About 95 percent of the solid waste generated in domestic vanadium mining operations consists of overburden material containing soil, clay, and various minerals associated with the ore body, such as quartz and feldspar. The remaining 5 percent consists of waste rock containing minerals such as illite, geothite, montroselite, pyroxene, quartz, feldspar, and kaolinite.

Concentrating Operations Wastes.

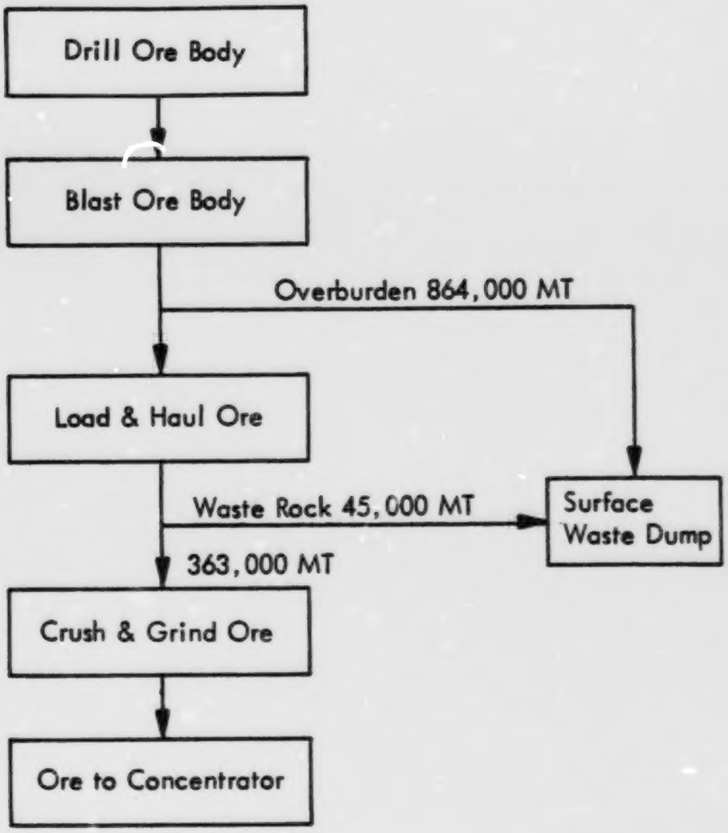
Roast Leach Process for Vanadium. As mentioned, there is only one production facility in the United States for the mining and concentrating of vanadium ore. As shown in the simplified process flow diagram (Figure 27), the basic mill circuit consists essentially of the following process steps:

1. Drying and grinding of the ore.

2. Blending the fine ore with sodium chloride and extruding the blended mixture to form agglomerated feed material for roasting.

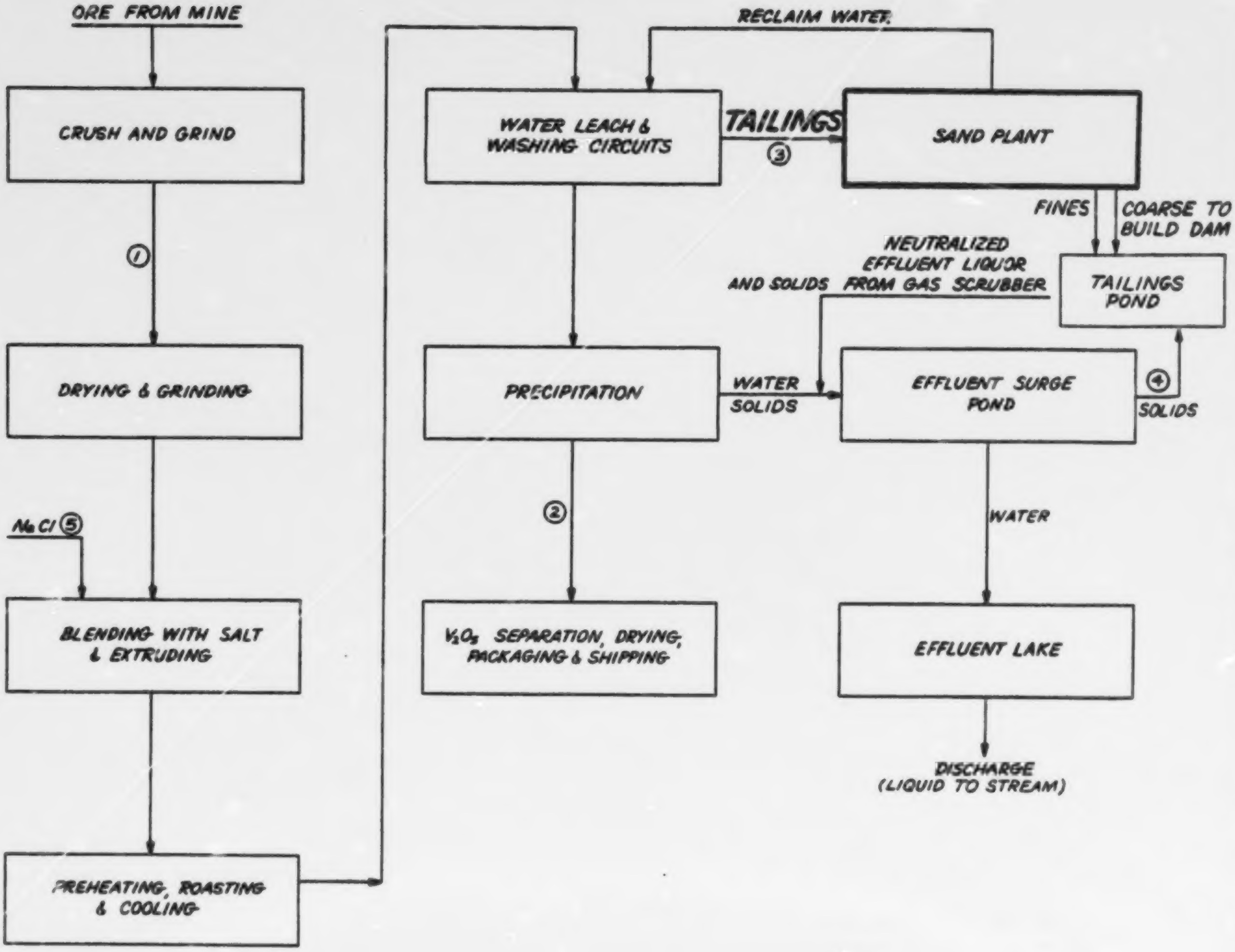
3. Roasting the mixture to yield a material containing water-soluble vanadium values.

4. Continuous water leaching, with warm water at about 45 C, at atmospheric pressure, and a pH of 6 to 8, of the roast in tanks to dissolve the contained vanadium. This leaching process requires several hours to complete.


Drill Ore Body
Blast Ore Body
Overburden 864,000 MT
Load & Haul Ore
Waste Rock 45,000 MT
Surface Waste Dump
363,000 MT
Crush & Grind Ore
Ore to Concentrator

Source: Reference 36.

Figure 26. Vanadium mining.


ORE FROM MINE
CRUSH AND GRIND
①
DRYING & GRINDING
NaCl ⑤
BLENDING WITH SALT & EXTRUDING
PREHEATING, ROASTING & COOLING
RECLAIM WATER
WATER LEACH & WASHING CIRCUITS
TAILINGS
③
SAND PLANT
FINES
COARSE TO BUILD DAM
NEUTRALIZED EFFLUENT LIQUOR AND SOLIDS FROM GAS SCRUBBER
TAILINGS POND
PRECIPITATION
WATER
SOLIDS
EFFLUENT SURGE POND
④
SOLIDS
②
WATER
V₂O₅ SEPARATION, DRYING, PACKAGING & SHIPPING
EFFLUENT LAKE
DISCHARGE (LIQUID TO STREAM)

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS	① ORE	② V_2O_5 CONC.	③ TAILINCS	④ TAILINGS	⑤ NaCl
Quantity, TPY	400.000	5.000	394.000 (Dry)	6.000	5.000
Metric TPY	362.874	4.536	357.431	5.443	4.536
V_2O_5 %	1.0	65 - 75	NA		
FeS_2 %	~5	0	NA		
Pb %	Present	0	NA		

Source: Reference 36.

Figure 27. Vanadium mining and processing.

TABLE 86

TOTAL PRODUCTION STATISTICS, BY STATE AND EPA REGION FOR SIC 1094 (URANIUM-VANADIUM* ORES) FOR 1974 (METRIC)

		Wastes generated (10^3 TPY)						Products, TPY			Ratio of:		
State	EPA Region	Ore mined	Over-burden	Waste rock	Tailings concentrator wastes	Total wastes	Concentrator wet waste (10^3 TPY)	V Conc.	Other metals	Total product	Total waste to ore	Total waste to product	Wet waste to ore
Arkansas	VI	363	862	45	368	1,275	1,587	4,536	0	4,536	3.52	281	4.37
National		363	862	45	368	1,275	1,587	4,536	0	4,536	3.52	281	4.37

Source: References 37 and 38.
* Arkansas is the only state producing vanadium ores.

TABLE 87

RATIO OF TOTAL WASTE ROCK-OVERBURDEN-CONCENTRATOR WASTE TO ORE MINED FOR SIC 1094 (URANIUM-VANADIUM* ORES) FOR 1974 (METRIC)

State	EPA Region	Ore (10^3 TPY)	Waste rock (10^3 TPY)	Ratio of total waste rock to ore	Overburden (10^3 TPY)	Ratio of total over-burden to ore	Concentrator wastes (10^3 TPY)	Ratio of total concen-trator waste to ore	Concentrator wet waste	Ratio of wet waste to ore
Arkansas	IV	363	45	0.12	862	2.38	368	1.01	1,587	4.37
National		363	45	0.12	862	2.38	368	1.01	1,587	4.37

Source: References 37 and 38.
* Arkansas is the only state producing vanadium ores.

5. Processing of the leach solution is accomplished by solvent extraction to produce a modified vanadium oxide. Tailings from this recovery system are sent to a tailings pond.

6. Separating, drying, packaging, and shipping of product vanadium oxide concentrate to a sister plant for further processing.

Vanadium ore obtained from a stockpile is dried and then ground. The drier and mill are equipped with dust collectors and wet scrubbers for treatment of off-gases before they are sent to a stack.

The ground ore is blended thoroughly with crushed salt ($NaCl$), and the blended material is then processed through an extruder to prepare agglomerated material suitable for use as feed for a roasting operation.

The agglomerated mixture of ore and salt is then preheated and roasted at about 850 C to convert the contained vanadium to a water-soluble sodium metavanadate ($NaVO_3$). The roaster is equipped with an off-gas scrubber. The vanadium is extracted by leaching with warm water (45 C) and precipitated as sodium hexavanadate, $Na_4V_6O_{17}$ (referred to as redcake) by the addition of sulfuric acid to adjust the pH to between 2 and 3.

The precipitated vanadium compound is separated from the liquid, dried, packaged, and shipped to a sister plant for further processing to a higher grade vanadium oxide product. The solids are sent to a tailings pond. The liquor is sent to an effluent pond where it is treated and discharged to a stream. This method of disposal has been judged satisfactory and complies with effluent limitations guidelines.

Mass Balance of Materials. In the Arkansas vanadium facility 362,874 MT (400,000 tons) per year of ore are processed in the concentrator by the roast-leach process. The total quantity of waste or solids discharged to a tailings pond is about 357,000 MT (394,000 tons) per year (dry weight). In addition, about 5,400 MT (6,000 tons) per year (dry weight) of waste solids from the effluent surge pond are sent to a tailings impoundment.

Description of Individual Waste Streams. The tailings discharged from the water-backing step of the concentrator process contain gangue material originally present in the feed ore. In 1974, there were 368,000 MT (405,651 tons) of tailings concentrator wastes. The ratio of total concentrator waste to ore was 1.01. There are no potentially hazardous wastes destined for land disposal from the mining and concentrating of vanadium ore.

Identification of Potentially Hazardous Wastes. There are no potentially hazardous wastes resulting from the treatment and disposal of wastes generated in the mining and concentrating of vanadium ores.

Waste Treatment and Disposal--Uranium

The potentially hazardous wastes which are land-disposed in the mining and concentrating of uranium ores are the mined waste rock located adjacent to the ore deposit and the concentrator tailings (a slurry of sand, slime, and liquid).

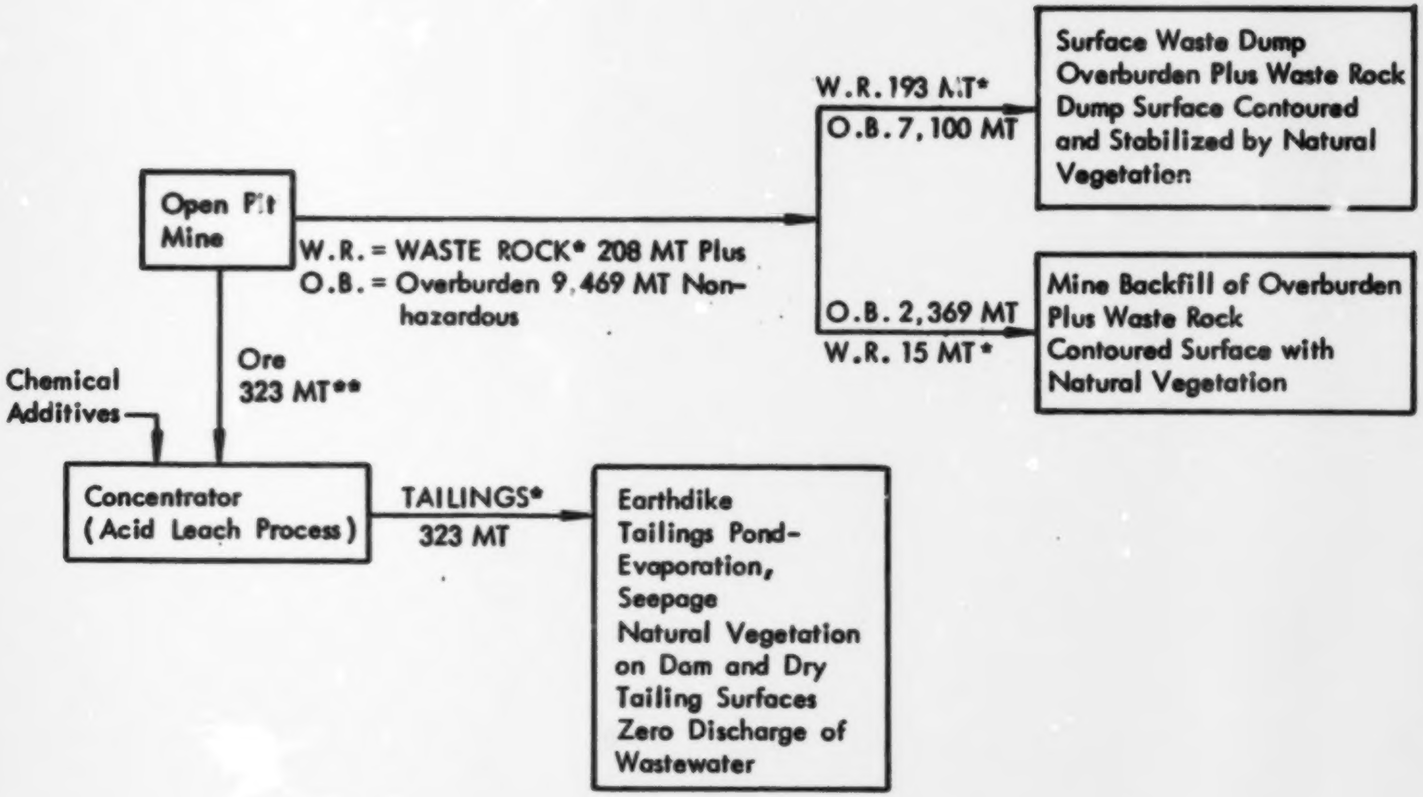
All domestic waste treatment and disposal operations practiced in the uranium mining and concentrating industry are conducted on the plant sites by the operating companies, and no waste materials are sold. A discussion of the treatment and disposal practices for these potentially hazardous wastes follows.

Open-Pit Mining Wastes. Open-pit mining accounted for about 58 percent (6,629 MT of U_3O_8) of the total uranium mine output in 1974.[26] The estimated total number of active open-pit mines during 1974 was 122.[27] Open-pit mining operations in Wyoming are typical.[28]

In 1974, the estimated national total of waste rock mined in open-pit uranium mine operations amounted to 2,026,000 MT (2,233,300 tons). This tonnage represented about 91 percent of the total waste rock mined during 1974 from both open-pit and underground mines

Waste Rock Treatment and Disposal. Waste rock, the only potentially hazardous waste involved in open-pit mining, is disposed of in either surface waste dumps or in a very few cases in mine backfilling operations. Most plants dispose of the waste rock along with the overburden waste (nonhazardous) which is also removed from the pits and comingled with the waste rock. Some mines segregate the waste rock from the overburden and store the waste rock in open surface piles for future use as a low grade feed ore; the estimated amount stored was 0.5 to 1 percent (33-66 MT of U_3O_8) of the total ore output of U_3O_8 values in 1974 for open-pit operations.

Levels I, II, and III Technology. Figure 28 is a flow diagram showing the representative waste disposal and treatment operations for Level I technology for disposal of potentially hazardous wastes based on 10,000 MT ore, waste rock, and overburden combined. This level of technology applies for about all of the operating mines. The waste rock, which represented on the average about 2.1 percent (208 MT) of the total material mined in the open-pit operation, is disposed of in either a surface dump or as backfill (along with comingled overburden which is a nonhazardous waste) in abandoned areas of open-pit mines. About 93 percent (193 MT) of the waste rock is permanently disposed of in surface dumps along with overburden material. The dump surface is contoured and stabilized by natural vegetation. The remainder of the waste rock (7 percent or 15 MT) is used along with overburden as backfill material for abandoned open-pit mine areas, following temporary storage in surface dumps.


Open Pit Mine
W.R. = WASTE ROCK* 208 MT Plus
O.B. = Overburden 9,469 MT Non-hazardous
W.R. 193 MT*
O.B. 7,100 MT
Surface Waste Dump Overburden Plus Waste Rock Dump Surface Contoured and Stabilized by Natural Vegetation
O.B. 2,369 MT
W.R. 15 MT*
Mine Backfill of Overburden Plus Waste Rock Contoured Surface with Natural Vegetation
Chemical Additives
Ore
323 MT**
Concentrator (Acid Leach Process)
TAILINGS*
323 MT
Earthdike
Tailings Pond-Evaporation, Seepage
Natural Vegetation on Dam and Dry Tailing Surfaces
Zero Discharge of Wastewater

Source: References 36 and 37.

Legend:

*Potentially hazardous waste.

**The numbers represent the estimated weighted average distribution of total materials mined, based on 10,000 metric tons.

Figure 28. Level I technology for treatment and disposal of potentially hazardous wastes in open-pit uranium ore mining and in ore concentrator operations.

In Level I technology, the surface waste dump method for disposal of a mixture of waste rock and overburden is not considered to be adequate for environmental protection. Under these conditions, the exposed surface areas of the pile not stabilized by natural vegetation are vulnerable to erosion by wind and water which could cause pollution problems.

The disposal method of using the waste rock and overburden mixture for mine backfilling operations is also considered to be unsatisfactory as a means of environmental protection because of potential erosion problems.

No significant effects on the future adequacy of Level I technology are anticipated as a result of changes in the volume and composition of wastes as new air and water pollution control facilities are installed or existing facilities are improved to meet more stringent control standards. It is unlikely that there will be any significant change in the volume or composition of waste rock due to air and water pollution enforcement actions.

Level I technology is adaptable for use with any domestic open-pit uranium mining operation. Either mode or both modes of disposal (i.e., surface waste dump or mine backfill) can be readily adapted at a given mine, and the deciding factor is usually the economics involved in the disposal operation. Backfilling is more expensive than disposal on a surface pile.

Energy requirements differ considerably between surface dump disposal and mine backfilling; the latter method consumes by far the greater amount of energy. Except for the energy requirements for waste disposal, there are no significant nonland related impacts involved in Level I technology.

Waste rock contains higher than background levels of radioactive materials consisting of uranium and its radioactive decay products (e.g., radium, thorium, etc.). These radioactive materials are the principal environmental hazards in the waste.

Factors which affect the potential hazards presented by the waste include the prevailing climate and the mode of disposal. For example, very windy locations pose an increased hazard that windborne particles of tailing solids may be carried outside the perimeter of the mining area.

The potential problems with Level I control technology center around the risk of erosion (i.e., wind and rainwater transport) of radioactive particles of waste rock. This risk applies particularly to the surface waste piles and, to a reduced degree, to backfilled areas. In instances where the waste rock is covered by a thick layer of overburden, this hazard probably ranges from negligible to very low. However, in cases where the waste rock is present on exposed surfaces of waste piles, and therefore is vulnerable to erosion, the environmental hazard is significant. In some geographic areas which have low rainfall levels (e.g., arid states such as New Mexico), the annual rainfall is insufficient to support natural vegetation on the surface piles. There is considerably less risk of erosion in the northwestern mining states (e.g., Wyoming and Washington), where natural vegetation is more easily maintained. Because most of the waste rock consists of large lumps and coarse particles, the potential hazard of windblown particulates (i.e., small particles) probably ranges from low to negligible depending upon climatic factors such as prevalence and velocity of winds in a given locality. Areas of high annual precipitation have a relatively high risk of water erosion.

Level I technology can be applied at a mine site without any significant implementation time requirements or equipment purchases. Regular mining equipment is used in the waste disposal operations. This technology is not expected to have a significant impact upon noise pollution problems or to eliminate all potential problems concerning environmental pollution. A potential intermedia pollution problem, which has not been thoroughly investigated, involves the possibility of surface water pollution by airborne contaminants transported off the plant property.

It should be especially noted that Level I technology contains no provisions for minimizing the dissemination of potentially hazardous wastes into air and water during the active lifetime of a waste dump, i.e., during the period of time (which may be several years) when the dump surface is being built up to the final contour which can then be allowed to stabilize with natural vegetation.

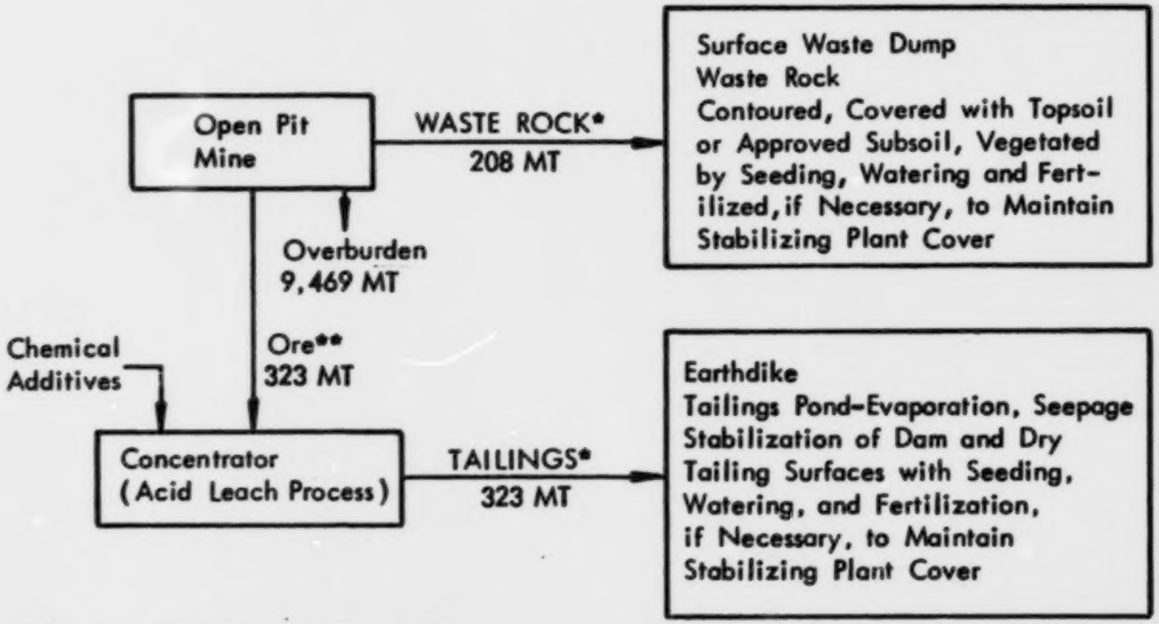
In Level I technology, monitoring is desirable to determine whether potentially harmful pollutants are entering the environment, and if so, to what extent. This activity should include periodic chemical analysis of samples of air, soil, vegetation, and surface water at the perimeter and near the plant site. In conjunction with these tests, natural background levels of radioactivity would be useful for comparison with the test samples.

Level II technology for disposal of potentially hazardous materials, which is also Level III technology, is practiced by about 10 percent of open-pit mines in areas where the state reclamation laws require application of advanced technology for waste disposal.

A flow diagram showing the Level II/III technology is presented in Figure 29. In this disposal method, waste rock segregated from overburden is deposited in a surface dump. After the dump becomes inactive, it is contoured, covered with a layer of compacted topsoil or approved subsoil, vegetated by seeding, and fertilized if necessary, to maintain a stabilizing plant cover which minimizes the erosion of the waste rock by wind and runoff water. The depth of soil cover used varies from one plant locality to another, depending on factors such as inherent differences in climate, topography, and availability of soil. No data were found which establish the required thickness of topsoil in different localities for long-term stabilization.

The period of greatest environmental risk in Level II/III technology is the active lifetime of the waste dump. During this period, which may under current practice last several years, the surface of the dump is in a state of flux, and is susceptible to water/wind erosion, and thus transport of potentially hazardous waste into water and the atmosphere. This risk can be reduced by shortening the effective active lifetime of the dump, i.e., by constructing a series of small dumps rather than a single large one or by periodically finishing and stabilizing segments of a large dump. Another alternative, particularly applicable to open-pit mines, consists of periodically covering layers of potentially hazardous waste rock with layers of nonhazardous overburden, i.e., operating the dump in a manner similar to a sanitary landfill in which each day's fill is covered with soil. The Level II/III technology will in the overall sense be adequate only if measures such as the above are, as appropriate in a specific mining operation, conscientiously implemented. The preferred measure is periodic covering of potentially hazardous waste rock with nonhazardous overburden or with available on-site soil materials. Added costs are expected to be essentially zero when overburden is abundantly available and must in any event be disposed in waste dumps. When overburden is not available, there will be an added expense associated with costs of moving and emplacing protective covers, or with scheduling the dump construction/stabilization protocol to reduce the effective active lifetime of the dump. The added cost has been estimated, in the following section on costs, for one assumed case in which an annual accumulation of wastes is covered with on-site generated cover (not overburden).

In arid regions (e.g., New Mexico) where watering would be required on a continuing basis to support vegetation, and where water is expensive, an acceptable alternate control method is available. This alternate method consists of contouring the surface of the inactive waste pile to conform to the landscape and then applying a layer of gravel or crushed rock onto the surface as a stabilizing cover.


Open Pit Mine
WASTE ROCK*
208 MT
Surface Waste Dump
Waste Rock
Contoured, Covered with Topsoil or Approved Subsoil, Vegetated by Seeding, Watering and Fertilized, if Necessary, to Maintain Stabilizing Plant Cover
Overburden
9,469 MT
Chemical Additives
Ore**
323 MT
Concentrator
(Acid Leach Process)
TAILINGS*
323 MT
Earthdike
Tailings Pond-Evaporation, Seepage
Stabilization of Dam and Dry Tailing Surfaces with Seeding, Watering, and Fertilization, if Necessary, to Maintain Stabilizing Plant Cover

Source: References 36 and 37.

Legend:
*Potentially hazardous waste.
**The numbers represent the estimated weighted average distribution of total materials mined, based on 10,000 metric tons.

Figure 29. Level II/Level III technology for treatment and disposal of potentially hazardous wastes in open-pit uranium ore mining and in ore concentrator operations.

Several mines segregate waste rock from the overburden during mining operations and store the waste rock in surface piles for future use as a low-grade feed ore; for 1974, the estimated amount stored was 0.5 to 1 percent (33-66 MT of U_3O_8) of the total output in U_3O_8 values for open-pit mines.

The Level II/III control technology, if conscientiously applied and maintained, should provide adequate protection against environmental pollution. The risks associated with this technology are concerned with the application of sufficient soil cover and seeded vegetation to ensure a uniform and complete plant cover which thoroughly stabilizes the surface and eliminates erosion problems, and with the care taken to minimize erosion when the dump surface is still in the active stage. For example, an inadequate depth of original soil cover on the waste rock pile could result, in very windy areas, in the soil being scoured off before the plant growth is able to stabilize the surface. The required depth of topsoil or subsoil varies from one locality to another depending on climatic conditions, availability of soil, and other factors.

No significant changes in volume or composition of the waste rock are expected as a result of future air and water pollution enforcement actions. Also, the new pollution control measures would probably not adversely affect the efficiency of Level II/III technology. All existing installations could be brought up to Level II/III technology without additional equipment or any serious retrofit problems.

With the exception of increased energy and chemical requirements (for applying soil covering and fertilizing), no significant nonland-related environmental impact is anticipated as a result of the implementation of this technology.

The technical discussion of the radioactive properties of waste rock given for Level I technology also applies here. Factors affecting the hazards presented by the wastes include climatic conditions and operating care exercised in maintaining a uniform and complete cover of plant growth over the entire surface of the waste rock pile. The principal problems concerned with Level II/III technology are the potential difficulties to be encountered in maintaining an adequate growth of vegetation cover on waste dumps in certain arid states where the climatic conditions are not amenable to revegetation as a reclamation method. For example, in some uranium ore processing areas (e.g., New Mexico), the annual rainfall is insufficient to sustain some seeded vegetation. As stated earlier, in arid regions of this type, it is believed that a suitable alternative procedure would involve depositing a covering layer of crushed rock or gravel upon the contoured surface of waste dumps as a means for minimizing or possibly eliminating erosion of waste rock by wind and water runoff. Such an alternative method should be completely effective in eliminating any windborne transport of waste rock particles off of the plant site.

No significant amount of implementation time would be required to apply the Level II/III technology, which can be largely accomplished using regular mining and concentrating equipment.

For Level II/III technology, monitoring is desirable to establish that no hazardous pollutants enter the environment because of wind and water erosion of waste piles. This monitoring should provide for periodic sampling and chemical analysis of air, soil, vegetation, and surface water at the plant perimeter and in the immediate plant vicinity.

Underground Mining Wastes.

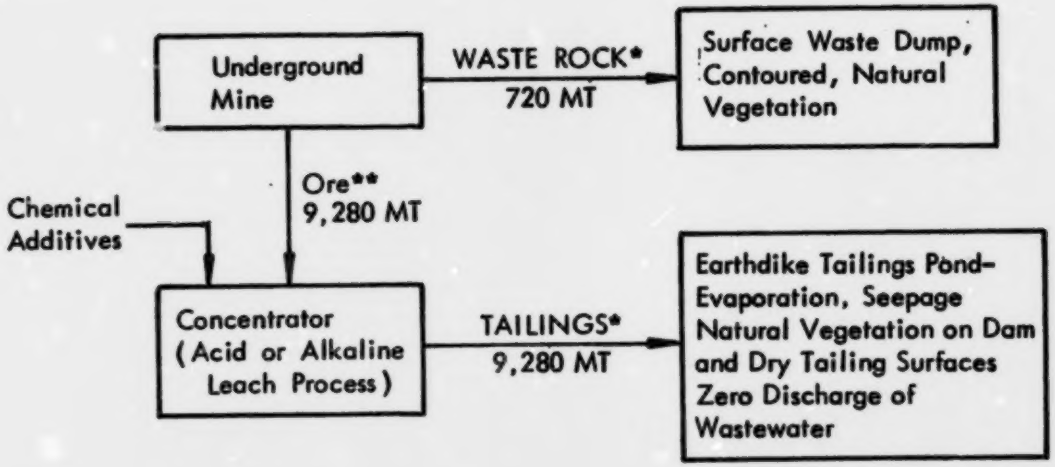
Waste Rock Treatment and Disposal. Underground mining represented about 40 percent (4,572 MT of U_3O_8) of the total uranium mine output in 1974.[26] The estimated total number of active underground mines during 1974 was 33.[27] Underground uranium mining operations in New Mexico are considered to be typical.

Waste rock is the principal potentially hazardous waste generated in underground mining. Mine water containing some dissolved uranium compounds is also potentially hazardous. Generally, this water is treated to recover the uranium values or is used in a concentrator process. Some waste rock is usually left in the mine. Some mines store the waste rock in surface piles for use as a low-grade feed ore; for 1975, the estimated amount stored was 0.5 to 1 percent (23-46 MT of U_3O_8) of the total ore output for underground mines.

The disposed or stored waste rock contains above background levels of potentially hazardous radioactive materials such as uranium and its radioactive decay products. A description of these radioactive properties is given in the previous section on open-pit mines. Both the prevailing climate and the topography in the disposal area are critical factors affecting the hazard presented by the waste rock.

Levels I, II, and III Technology. Level I technology for disposal of potentially hazardous waste (Figure 30) in underground mining of uranium ore consists of depositing waste rock onto open surface dumps, terracing the surface to conform to the natural landscape, and relying on natural vegetation to provide a cover of native plants or grass. This level of technology is practiced by almost all of the operating plants.

The Level I technology is not considered to be environmentally adequate because exposed surfaces, not protected by natural vegetation on the dump, are subject to wind and water erosion of the waste rock, which could result in environmental pollution problems.


Underground Mine
WASTE ROCK*
720 MT
Surface Waste Dump, Contoured, Natural Vegetation
Ore**
9,280 MT
Chemical Additives
Concentrator (Acid or Alkaline Leach Process)
TAILINGS*
9,280 MT
Earthdike Tailings Pond-Evaporation, Seepage Natural Vegetation on Dam and Dry Tailing Surfaces Zero Discharge of Wastewater

Source: References 36 and 37.

Legend:
*Potentially hazardous wastes.
**The numbers represent the estimated weighted average distribution of total materials mined, based on 10,000 metric tons.

Figure 30. Level I technology for treatment and disposal of potentially hazardous wastes in underground uranium ore mining and in ore concentrator operations.

The future installation of new air and water pollution control facilities is not expected to materially affect either the volume or the composition of the land-disposed wastes.

Since all mines are applying at least Level I technology, there are no problems of retrofitting this technology to existing installations. With the exception of the energy requirements for contouring the waste dumps, there are no significant nonland-related environmental impacts. Level I technology can be implemented readily with little or no time delay at any new mine site. Application of this technology does not significantly affect the existing noise pollution problems at a given site.

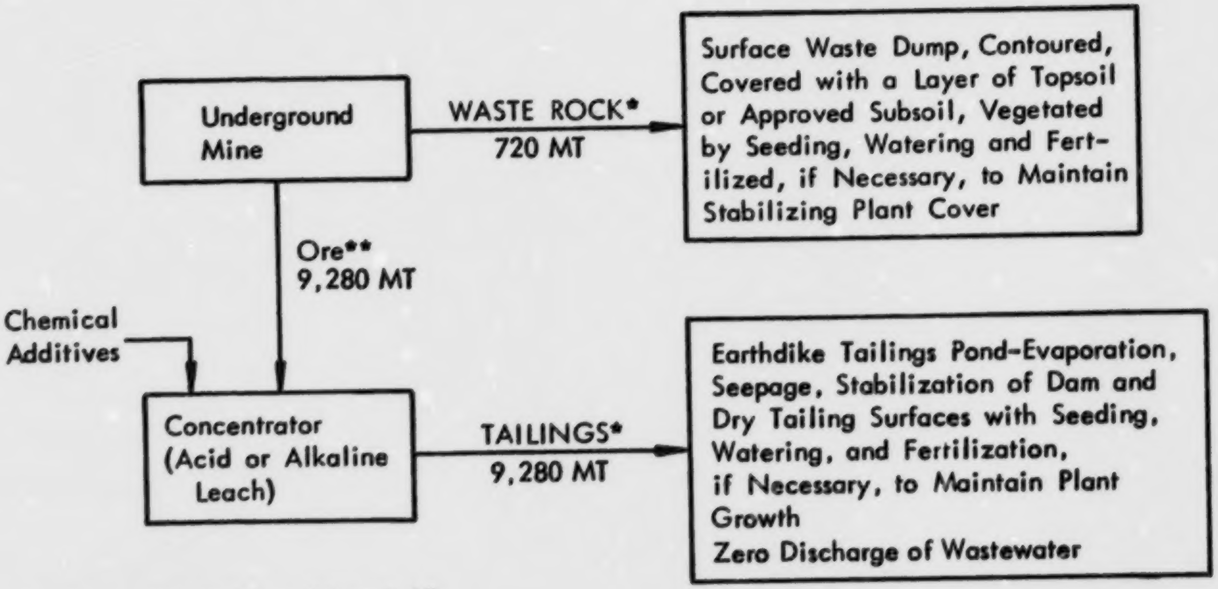
Monitoring activities comprised of periodic sampling and analysis of samples of air, soil, vegetation, and surface water at the perimeter of the plant site are desirable, particularly since it has been concluded that Level I technology may not adequately protect the environment.

As shown in Figure 31, Level II technology, which is also considered to be Level III technology, includes depositing waste rock from underground mines on open surface dumps, contouring the surface to match the natural landscape, spreading a layer of topsoil or approved subsoil onto the contoured surface, and revegetating by seeding for stabilization against erosion. Fertilization and watering are carried out, if necessary, to maintain plant growth. It is estimated that about 10 percent of the mines use this technology on inactive dumps. The Level II/III technology described above for inactive dumps must include measures designed to protect the dump surface during the active stage as discussed earlier for open-pit mining wastes. These measures include providing interim covers of nonhazardous soil materials or scheduling dump construction so that dump surfaces are in the active stage for relatively short periods of time.

In arid regions where rainfall is inadequate to support plant growth and irrigation is expensive, an alternative stabilization method, as for open-pit mine wastes, which involves application of a layer of crushed rock to the surface of contoured piles, is considered by the authors of this report to be an acceptable and effective procedure.

The Level II/III control technology should be environmentally adequate, if it is conscientiously applied. Risks associated with this control method are concerned with the application of sufficient soil cover and revegetation (or crushed rock) to provide a complete stabilizing cover which prevents problems with erosion of the waste rock.

The small anticipated changes in the volume and composition of the waste rock as future air and water pollution control facilities are installed would not adversely affect the adequacy of the Level II/III technology.


Underground Mine
WASTE ROCK*
720 MT
Surface Waste Dump, Contoured, Covered with a Layer of Topsoil or Approved Subsoil, Vegetated by Seeding, Watering and Fertilized, if Necessary, to Maintain Stabilizing Plant Cover
Ore**
9,280 MT
Chemical Additives
Concentrator (Acid or Alkaline Leach)
TAILINGS*
9,280 MT
Earthdike Tailings Pond-Evaporation, Seepage, Stabilization of Dam and Dry Tailing Surfaces with Seeding, Watering, and Fertilization, if Necessary, to Maintain Plant Growth
Zero Discharge of Wastewater

Source: References 36 and 37.

Legend:
*Potentially hazardous wastes.
**The numbers represent the estimated weighted average distribution of total materials mined based on 10,000 metric tons.

Figure 31. Level II/III technology for treatment and disposal of potentially hazardous wastes in underground uranium ore mining and in ore concentrator operations.

No problems are anticipated in retrofitting the Level II/III technology to installations having Level I technology, aside from the increased cost for the improved control system. The nonland-related environmental impact would be primarily the increased energy and fertilizer requirements for Level II/III operation.

The Level II/III technology could be planned and implemented at any Level I site or new facility within 6 to 12 months. It is not anticipated that this technology would have any significant impact on the existing noise pollution problems at a given site.

A discussion of the potentially hazardous constituents in the waste rock is given in the description of Level I technology for open-pit mining. Factors affecting the hazards posed by the waste rock include climatic factors (e.g., wind and rain storms and other severe weather) and the degree and continuity of operating care used in maintaining an adequate stabilizing cover on the waste dump. Thus, the reliability of the environmental protection would depend in large measure on continuing maintenance of properly stabilized waste dumps and on the adequacy of measures adopted for protecting surfaces of active dumps.

The activities necessary to adequately monitor the disposal operations involve periodic collection and analysis of samples of air, soil, vegetation, and surface water at the plant perimeter and in the vicinity of the mine site.

Concentrator Operation Wastes.

Waste Treatment and Disposal. The potentially hazardous land-disposed wastes in the uranium ore concentrator operations are the tailings which consist of a slurry of sand, slime, and liquid. Large quantities of potentially hazardous tailings wastes have been generated by domestic plants. For example, from 1948 through 1972, the domestic uranium concentrator industry processed 93,510,300 MT (103,078,000 tons) of ore containing 221,100 MT (243,700 tons) of U_3O_8.[28/] This operation represents an accumulation of about 93 million metric tons (102.5 million tons) of solid waste (tailings).[28/]

All uranium ore concentrator plants in the United States use the same method for disposal of tailings, i.e., the use of a tailings pond (impoundment basin) for containment of the solid wastes. Liquid disposal is accomplished by natural evaporation and seepage from the pond. No evidence has been found that pond seepage migrates beyond the plant perimeter or causes any pollution problems in current operations.[28/] The size of these tailings ponds varies from a few hectares to over 100 hectares (250 acres), and from 1 to 10 ponds may be used at individual concentrator sites.[29/]

The tailings slurry is commonly spigotted into the impoundment area from a peripheral pipeline. The dam height is increased periodically as needed by application of coarse sand separated from the tailings slurry by a portable cyclone; the separated slimes are discharged away from the embankment areas into the pond. The tailings pond construction and operation are basically the same for all types of concentrator plants.

Of the 16 active domestic concentrator plants in 1974, 12 plants used an acid leach process and two plants used a similar dual process which involves both an acid leach and an alkaline leach step. These concentrators represented about 85 percent (10,143 MT) of the total production of U_3O_8 concentrate and generated 80 to 90 percent (4,572,000-5,144,000 dry MT) of the total concentrator tailings waste during 1974.

In concentrating processes using the acid leach process, the average quantity of solid tailings waste is estimated to be 1 MT/MT of uranium feed ore consumed in the process.[28] Also, the total quantity of waste solution accompanying the solids (consisting of sands and slimes) in the tailings slurry is estimated as 1.5 MT/MT of the feed ore processed.[28] Sears et al.[28] estimate that the slime fraction (< 200 mesh particles) contains 85 percent of the insoluble radioactive materials originally in the feed ore.[28] The weight ratio of sands to slimes in the dry solids fraction of tailings is 2:33 to 1. Detailed data on the volume and composition of the tailings for a model plant using the acid leach process are shown in Figure 24, p. 181. Potentially hazardous radioactive materials are contained in both fractions of the solid waste and in the liquid waste.

During 1974, two domestic concentrator plants used the alkaline leach process for production of uranium concentrate. These plants contributed about 14 percent (1,651 MT) of the total concentrate production and generated about 10 to 20 percent (572,000-1,143,000 dry MT) of the total tailings in 1974.

For concentrator plants using the alkaline leach process, the average quantity of solid tailings waste is estimated to be 1 MT/MT of feed ore processed.[28] The total waste solution contained in the tailings slurry is about 1.05 MT/MT of the feed ore consumed.[28] The slime fraction contains about 85 percent of the insoluble radioactive materials originally present in the feed ore.[28] The dry solids fraction of tailings consists of equal parts by weight of sands and slimes. Detailed data on the volume and composition of the tailings for a model alkaline leach plant are given in Figure 25, p. 187. All fractions of the tailings waste contain potentially hazardous radioactive materials.

Level I Technology. The Level I technology presented in this report deals with active tailings ponds. For a discussion of methodologies applicable to inactive tailings ponds, the reader is referred to a cost benefit study[27] which encompasses inactive tailings ponds.

Level I technology for the waste treatment and disposal of the potentially hazardous wastes from uranium concentrating processes involves discharging all the tailings waste from the concentrator to an earthen-dike tailings pond as shown in Figure 28, p. 203. The residual wastes finally disposed in the tailings ponds include the sand, slime, and liquid contained in the tailings slurry discharged to the tailings pond. Natural evaporation and seepage are the principal means for disposal of liquid contained in the tailings slurry.[28] In some plants, a portion of the settled clear liquor in the tailings pond is recycled to the concentrator process, especially in areas where water is in short supply. Waste liquor from concentrators using an acid leach contain various heavy metals and are generally unsuitable for recycling without special treatment. The wastewaters from alkaline leaching are more amenable for reuse and are, therefore, commonly recycled in part to the process.[30]

All of the domestic concentrator plants use Level I disposal technology. Most of the concentrators (about 80-90 percent) have zero discharge of liquid effluent from their tailings ponds. The other plants carefully treat and monitor the effluent liquor to remove all potentially hazardous pollutants prior to discharge into water courses.[30]

Stabilization of the surfaces of the impoundment dam against erosion is accomplished in Level I technology by growth of natural vegetation. In some localities, natural plant growth provides good surface stabilization. Some natural vegetation also occurs on the exposed dry surface areas of the tailings ponds, and this aids in stabilization of the surface against wind erosion. In some arid localities, however, the natural rainfall is inadequate to support a suitable cover of vegetation. Natural vegetation, thus, is not in the general sense considered to be adequate environmental protection.

In the Level I technology for concentrator wastes, plants which have zero discharge of liquid effluent (80-90 percent of total Level I plants) are considered to be fully adequate from the standpoint of protection against pollution by wastewater. Also, the practice in some Level I plants of discharging some treated wastewater (i.e., treatment to remove potentially hazardous materials) is considered to be environmentally adequate; however, this practice involves risk of careless handling which could result in improperly treated effluent and pollution of surface streams. Therefore, zero discharge is considered to be a safer method of waste treatment and disposal than treatment and discharge of wastewater.

The only risk of environmental pollution in the Level I technology is concerned with the possibility of windborne losses of tailings from exposed areas of the tailings dam and dry beaches in the tailings pond. Depending upon climatic conditions and topography in a given concentrator locality, there is a significant risk that potentially hazardous tailings solids can be carried beyond the limits of a plant site by prevailing wind conditions. In some geographic areas (e.g., Colorado), natural vegetation is reported to be an effective method for stabilizing the surface of tailings dams. However, in some arid climates, natural vegetation is not adequately supported by the existing rainfall in the region, and insufficient natural groundcover is present to prevent erosion.

The future installation of new air and water pollution control or improvement of the efficiencies of existing facilities is not expected to have a significant impact on either the volume or composition of concentrator plant wastes. These changes would probably have a negligible effect on the volume of wastes and their composition since nearly all of the existing waste materials are presently being collected and disposed. Therefore, the adequacy of the Level I technology would be essentially unaffected by future changes in air and water pollution control equipment. There are no problems of retrofit of waste treatment and disposal technology since all existing plants utilize at least the Level I technology. Level I technology can be applied at a concentrator site well within the time required to plan and construct the uranium ore concentrator facility. The technology for construction and operation of tailings ponds has been well established in the mining industry for many years; implementation of this technology does not involve any special problems in engineering studies, equipment, delivery, and startup.

The nonland-related environmental impact is concerned principally with the energy requirement for installation and operation of the tailings ponds. This requirement includes the energy required to construct the initial impoundment dam to pump the tailings slurry from the concentrator plant to the tailings pond and to periodically build up the height of the dam. This level of technology does not create any significant noise pollution problems. However, it is conceivable that this technology could, in some instances, cause problems concerning air and water pollution in the immediate vicinity of the operating site. For example, windborne particles of tailings waste could be transported beyond the perimeter of the plant. A possible, but unproven, intermedia problem is concerned with the possible pollution of surface water by contaminants transported from a tailings pond area (e.g., dry tailings areas or exposed dam surfaces).

Regularly scheduled monitoring of the treatment and disposal facility operations is desirable. This activity would comprise periodic chemical analysis of air, soil, vegetation, surface water, and groundwater at the plant perimeter and in the plant vicinity. These monitoring tests would serve to establish whether any tailings waste was being transported by wind or water erosion into the environment. Data on natural background levels of radiations would be needed for comparison with the test sample data.

Level II/III Technology. Level II technology for disposal of potentially hazardous wastes which is also considered to be Level III technology consists of tailings ponds having zero discharge of effluent and special treatment to minimize erosion losses of tailings. The exposed surfaces on the impoundment dam and dry tailings in the pond area are stabilized with seeded vegetation, fertilized, and watered, if necessary, to maintain plant growth. About 10 percent of the concentrator plants use the Level II/III disposal technology. The impoundment dam is periodically built up with separated coarse tailings sand by the same method as that described previously for the Level I technology. Since all plant operations at this level of technology have zero discharge of effluent wastewater from the ponds, there is no risk of environmental pollution due to discharge of any liquid waste.

Stabilization with seeded vegetation of the exposed surfaces of the tailings dam and the dry tailings beaches serves to minimize the risk of windborne transport of tailings dust beyond the perimeters of the plant site. The conscientious application and maintenance of vegetation cover should serve to adequately protect the environment against pollution by wind or water erosion of finely divided tailings particles. Improper maintenance of a vegetative cover could result in significant risks of airborne pollution, particularly in windy locations.

It is not anticipated that the installation of new or improved air and water pollution control facilities would significantly affect the volume or composition of wastes generated by the uranium ore concentrator plants. Therefore, it is believed that the future adequacy of Level II/III technology will remain essentially unchanged after new control facilities are installed.

With one exception, there are no serious problems of retrofitting the Level II/III technology to existing installations having the Level I technology. Level I plants which own insufficient land to develop the required size of tailings pond may have to continue to treat and discharge a portion of their tailings wastewater. For Level I plants having no liquid discharge from the tailings ponds, the changeover to Level II/III is relatively easy, i.e., a change from the use of natural vegetation to seeded vegetation as a means of stabilizing the surface of the active tailings ponds.

The principal nonland-related environmental impact of Level II/III technology is the energy requirement for installing and operating the tailings ponds and the chemicals requirement for fertilization.

Under certain conditions, Level II/III technology could be rendered partially ineffective in regard to possible air pollution problems. If the seeded vegetative cover on the dam and the dry areas of the active tailings pond are not continuously maintained by periodic reseeding, fertilizing, and watering as necessary, then barren areas would develop, and air pollution problems could result.

Some treatment of liquid waste discharge would be required in the event that the volume of concentrator operations was greatly increased without a corresponding increase in the available tailings pond disposal area.

The estimated implementation time to convert a Level I installation to Level II/III including engineering studies, equipment delivery, and installation for applications and maintenance of seed vegetative cover is 6 months to 1 year. The application of Level II/III technology at a given site would not create any new problems concerning air, water, and noise pollution.

The activities necessary to properly monitor the waste disposal operations involve a regularly scheduled system for collection and analysis of test samples of air, soil, vegetation, and surface water at the plant perimeter and the immediate vicinity of the site.

Cost Analysis--Uranium-Radium-Vanadium Ores

Disposal Costs--Open-Pit Mine Wastes.

The rationale and assumptions used in developing a representative model for open-pit uranium mining operations are explained on page 172 under "Mass Balances of Materials." The mass balance data for a representative plant model and for waste treatment and disposal practices of open-pit mining operations are shown in Figures 24 (p. 181), 28 (p. 203), and 29 (p. 207).

Waste Treatment Costs - Surface Waste Dump, Level I Technology. The waste treatment costs of uranium mining and concentrating will be broken into two major categories. First, the costs of depositing overburden and waste rock in surface waste dumps and mine backfill will be examined. Next, the cost of treating the tailings from the ore concentrator will be assessed.

The representative open-pit uranium mine creates 1.942 by 10^7 MT of overburden per year of which 1.46 by 10^7 MT (75 percent) is deposited in the surface waste dump. The rest is used as mine backfill. The mine also produces 425,700 MT of waste rock per year. Approximately 395,000 MT (93 percent) of the waste rock is deposited in the surface waste dump. The Level I disposal technology does not separate overburden and potentially hazardous waste rock.

The first step in this calculation is to determine the capital costs of the surface dump. A total of 1.5 by 10^7 MT/year of waste and overburden are deposited on the dump. We will assume a bulk density of both materials equal to 1.96 MT/m^3 (1.5 MT/yd^3).[33] Therefore, 7.62 by 10^6 m^3/year (9.97 by 10^6 yd^3/year) are deposited. Over 20 years, the dump will contain 1.52 by 10^8 m^3 (1.99 by 10^8 yd^3) of rock. If the dump is 7.62 m deep (25 ft), the land required would be 1.99 by 10^7 m^2 or 1,995 hectares (4,929 acres). Assuming a land value of $12,350/hectare ($5,000/acre) in 1973 dollars, the total land cost will be approximately $24.05 by 10^6 for the surface dump.

The $24 million cost of the surface dump can enter the capital cost calculation in at least three distinct ways. First, one can assume all 1,995 hectares are bought during the 1st year of the analysis. This land may have the same value at the end of the period of analysis (20 years) or have a reduced value. On the other hand, the land can be purchased incrementally over the period. Approximately 100 hectares (250 acres) are needed per year. At $12,350/hectare, the annual acquisition cost of land would be $1.2 by 10^6. Again, the land may have the same or reduced value at the end of the period. Last, the land may be considered as part of the conventional costs of operating the mine and not attributable only to potentially hazardous waste disposal. In this case, the value of the land is not relevant to the cost calculation and should be omitted.

The answers obtained using these alternative approaches are quite different. The difference occurs because time is being discounted at a 10 percent rate. The present value of alternative capital cost outlays will therefore vary significantly. To give the broadest meaning to the results, the cost of waste disposal is calculated using each of the alternative capital cost outlay patterns. These alternatives "cases" are carried throughout the analysis. Other major cost assumptions are presented in Table 88.

The land cost calculations presented above and the other assumptions in Table 88 were used to estimate the cost of the surface waste dump for the disposal of uranium. Table 89 presents the results. The dollar costs in Table 89 are expressed as annual amounts, except where otherwise specified, expressed in fourth quarter (Q4) 1973 dollars. Table 89 indicates that the cost per metric ton of potentially hazardous waste disposal using a surface waste dump (Level I technology) can range from \$0.23 to \$0.003, depending on which method is used for evaluating land costs. Total industry costs associated with Level I technology would be \$0.5 million to \$8,000 or less than 0.5 percent of the total value added in mining uranium ores.

Waste Treatment Cost--Mine Backfill, Level I Technology. In addition to surface waste dumping, potentially hazardous waste materials can be disposed by mine backfilling. Level I technology for mine backfilling is a process which does not separate overburden and waste rock. This rock is temporarily piled relatively close to the pit and later trucked back to the edge. The material is contoured, and natural vegetation occurs. The cost of potentially hazardous waste disposal by mine backfilling (Level I technology) includes: (1) land rental or purchase; (2) hauling the material back to the pit; (3) contouring the surface.

As mentioned earlier, the representative open-pit uranium mine must dispose of 1.94 by 10^7 MT of overburden per year of which 4.86 by 10^6 MT (25 percent) is used as mine backfill. The mine also produces 4.257 by 10^5 MT of potentially hazardous waste rock per year, of which 30.7 by 10^3 MT (7 percent) are used as backfill. The total amount of mine backfill material equals 4.89 by 10^6 MT/year. Assuming a bulk density of 1.96 MT/m^3 (1.5 MT/yd^3), 2.49 by 10^6 m^3/year will be piled for later mine backfill. If the pile is 7.62 m deep (25 ft), the pile will cover 3.27 by 10^5 m^2/year (80.8 acres) or 32.7 hectares/year. We will further assume that the temporary pile is allowed to accumulate for 5 years before backfilling begins. Once backfilling begins, it is assumed to proceed at a rate greater than the rate that material is being added. In other words, the maximum land area needed is 163.5 hectares. At a price of \$12,350/hectare (\$5,000/acre) in 1973 dollars, the initial land cost would be \$2.02 by 10^6. Reselling the land after 20 years would yield a total rental cost of \$1.69 by 10^6. Applying the capital recovery factor (20 years at 10 percent) yields a levelized land rent of \$1.98 by 10^5/year.

TABLE 88

MAJOR ASSUMPTIONS OF COST ANALYSIS, OPEN-PIT URANIUM MINE--SURFACE WASTE DUMP--TECHNOLOGY LEVEL I

I. Capital Cost (other than land)

One bulldozer is assumed to work full-time at surface waste dump

a. Bulldozer (Cat-977) Capital cost = \$56,000*
Useful life = 5 years†

b. Present value calculations

Year purchased or replaced	Real dollar cost (1973)	Present value factor‡	Present value of cost
1	56,000	1.0	\$56,000
6	56,000	0.621	34,776
11	56,000	0.386	21,616
16	56,000	0.239	13,384
Total			\$125,776

Capital recovery factor (20 years at 10%) = 0.117
Levelized annual capital cost of bulldozer = \$14,716

II. Labor Cost

One bulldozer operator required at \$8.97/hr (1973 dollars, 40)§
Labor cost = \$8.97 x 8 hr/day x 260 days/year
= \$18,658/year

III. Supervision

Assumed to be 25% of labor cost**
Supervision = 0.25 x \$18,658/year = \$4,664

IV. Maintenance

Assumed to equal 6% of nominal cost of bulldozer per year††
Maintenance = 0.06 x \$56,000 = \$3,360/year

TABLE 88 (CONCLUDED)

V. Insurance and Taxes

Assumed to equal 2% of nominal capital cost (land and equipment)[‡‡]
Insurance and taxes = $0.02 \times (\$24.05 \times 10^6 + 56.0 \times 10^3) = \$4.82 \times 10^5$

VI. Energy and Power

Bulldozer assumed to use 60 gal. diesel fuel per day[‡‡]
Energy plus power cost = 260 days/year x 60 gal/day x \$0.50/gal[‡‡]
= \$7,800/year

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[*] U.S. Environmental Protection Agency. Assessment of industrial hazardous waste practices, inorganic chemical industry. Contract No. 68-01-2246, 1975. p. 7-33.

[†] U.S. Environmental Protection Agency. Assessment of industrial hazardous waste practices, inorganic chemical industry. Contract No. 68-01-2246, 1975. p. 7-33.

[‡] Assuming a 10% discount rate (r), the present value factor equals $\frac{1}{1 + r}^{i}$ where i = the relevant year 1 in column 1.

[§] U.S. Environmental Protection Agency. Assessment of industrial hazardous waste practices, inorganic chemical industry. Contract No. 68-01-2246, 1975. p. 7-5 and C-2.

[**] U.S. Environmental Protection Agency. Assessment of industrial hazardous waste practices, inorganic chemical industry. Contract No. 68-01-2246, 1975. p. 7-34.

[††] Gruber, G. I. Assessment of industrial hazardous waste practices, organic chemical pesticides and explosives industries. U.S. Environmental Protection Agency, Contract No. 68-01-2919, January 1976. p. 7-2.

[‡‡] U.S. Environmental Protection Agency. Assessment of industrial hazardous waste practices, inorganic chemical industry. Contract No. 68-01-2246, 1975. p. 7-34.

TABLE 89

DISPOSAL COSTS--POTENTIALLY HAZARDOUS WASTE FROM OPEN-PIT URANIUM MINING--SURFACE WASTE DUMP, TECHNOLOGY LEVEL I (EXPRESSED IN ANNUAL COSTS--1973 (Q4) DOLLARS)

	Case "A"*	Case "B"†	Case "C"‡	Case "D"§
Capital	$2.83 x 10^6	$2.36 x 10^6	$1.2 x 10^6	$1.47 x 10^4
Labor	$18,658	$18,658	$18,658	$18,658
Supervision	$4,664	$4,664	$4,664	$4,664
Maintenance	$3,360	$3,360	$3,360	$3,360
Materials	None	None	None	None
Insurance + taxes	$4.82 x 10^5	$4.82 x 10^5	$2.4 x 10^4	1,120
Energy + power	$7,800	$7,800	$7,800	$7,800
Total cost of surface dump	$3.34 x 10^6	$2.87 x 10^6	$1.25 x 10^6	$5.03 x 10^4
Percent of metric tons of hazardous waste rock (in total waste)	2.7	2.7	2.7	2.7
Cost of hazardous waste disposal	90,360	77,650	33,800	1,360
Metric tons of potentially hazardous waste to surface dump each year	395 x 10^3	395 x 10^3	395 x 10^3	395 x 10^3
Cost per metric ton of potentially hazardous waste in surface dump per year	$0.23	$0.20	$0.09	$0.003
Total cost if entire industry adopted ($$10^6$/yr)	$0.47	-	-	$0.008
Total cost as a percent of value added in mining	0.29%			0.0%

* Case "A" assumes all land purchased in Year 1 and has 0 value after 20 years.

† Case "B" assumes all land is purchased in Year 1 and is resold at the same value after 20 years.

‡ Case "C" assumes land is purchased yearly (247 acres) and has 0 value after 20 years.

§ Case "D" assumes land would be purchased even if no waste disposal was required and, therefore, land costs do not enter cost calculation.

Other capital equipment for mine backfilling includes bulldozers, front loaders, and trucks. The total amount of material to be hauled back to the open pit will be 4.98 by 10^7 m^3 (i.e., 2.49 by 10^6 m^3/year by 20 years). If mine backfilling begins after 5 years and continues until the end of the period of analysis (20 years), approximately 3.32 by 10^6 m^3 will have to be moved per year. We will assume 77-MT capacity (85 tons or 39.31 m^3) trucks will be used, and a round trip from the temporary pile to the edge of the pit takes 20 min. Therefore, 4.5 trucks will be needed on a full-time basis (24 hr/day, 260 days/year). In addition, one 7.6-m^3 (10 yd^3) front loader would be required full time. Finally, one bulldozer would be required 33 percent of the time. The cost and useful life of this equipment is shown in Table 90. Table 90 also presents the labor, operation, maintenance, taxes, and energy costs assumptions for the mine backfilling operation.

The assumption presented in Table 90 and the land costs discussed above were used to generate the estimates of the cost of disposal of potentially hazardous uranium wastes through mine backfilling (Table 91). The four cases of land valuation used in Table 91 are similar to the four cases presented in the discussion of disposal using surface waste dumps. However, Case "C" is slightly different. Because the mine backfill dump is only temporary, material will only accumulate for 5 years. For the next 15 years, the material is assumed to be backfilled to the mine at a faster rate than it is being placed on the pile. In Case "C", 32.7 hectares are assumed to be purchased each year for 5 years. The land is then held for 15 additional years and is assumed to have no resale value.

The results of the cost analysis show that the valuation of land is less important to the cost of mine backfill than in the case of surface waste dumping. The other capital equipment (trucks, loaders, and bulldozer) is a much more important cost component in the mine backfill operation. The associated labor supervision, maintenance, and energy also become more important. Table 91 shows that the cost per metric ton of potentially hazardous waste in mine backfilling is between \$0.21 and \$0.16 for the representative mine (Level I technology). These costs expressed as total cost to the industry represent \$0.023 to \$0.030 million per year or less than 0.5 percent of the value added in mining.

Waste Treatment Costs--Open-Pit Uranium Mine Waste, Levels II and III Technology.

Surface Waste Dump Disposal Costs--Levels II and III Technology. Disposal costs associated with Levels II and III technology differ from Level I technology in two important ways. First, the potentially hazardous waste rock from open-pit uranium mining is separated from overburden in Levels II and III technology. Therefore, a much smaller amount of material is deposited on the potentially hazardous waste dump. Second, the waste dump is contoured, covered with top soil (4 or 12 in. in the analysis), seeded, and fertilized, if necessary. The major cost assumptions used in the calculation are presented in Table 92.

TABLE 90

ASSUMPTIONS OF COST ANALYSIS, OPEN-PIT URANIUM MINE--MINE BACKFILLING--TECHNOLOGY LEVEL I

I. Capital Cost (except land)

A. Trucks

77.1 MT (85-ton) trucks are assumed to be used
Temporary pile is 1.6 km (1 mile) from edge of pit.
Hauling time equals 5 min to load, 6 min in transit (loaded) 3 min to dump, 4 min back to pile (unloaded) or ≈ 20 min/trip

Each truck hauls 2,832 cu meters/day (24 hr)
3.32×10^6 cu meters (4.34×10^6 cu yd) must be moved each year.
3.32×10^6/year ÷ 260 working days/year = 1.28×10^4 cu meters/day

Therefore, 4.5 trucks are required full time (24 hr/day and 260 days/year)

85-ton truck cost (in 1973 dollars) = $47,000*
Useful life = 5 years†

B. Front loaders

One caterpillar (Cat-992) with 7.6 cu meter capacity (10 cu yards) is required full time

Front end loader cost = $265,000 (1976 dollars)‡ or $203,000 (1973 dollars)§
Useful life = 5 years**

C. Bulldozer

From previous analysis it was assumed that one bulldozer (Cat-997) could contour 1.49×10^7 MT/year of overburden and waste rock. Therefore, the bulldozer is required 0.33 of the time for 4.89×10^6 metric tons/year.

Bulldozer cost (Cat-977) = $56,000 (1973 dollars)†
Useful life = 5 years‡

TABLE 90 (CONTINUED)

D. Present value calculation

Trucks - 4.5 at \$47,000 = \$2.115 x 10^5
Loader - 1.0 at \$203,000 = \$32.03 x 10^5
Bulldozer - 0.33 at \$56,000 = \$1.848 x 10^4
Total nominal cost (1973 dollars) = \$4.330 x 10^5

Year purchased or repaired	Nominal cost of equipment	Present value factor at 10%	Present value of cost
1	\$4.33 x 10^5	1.0	\$4.33 x 10^5
6	\$4.33 x 10^5	0.621	\$2.69 x 10^5
11	\$4.33 x 10^5	0.388	\$1.68 x 10^5
16	\$4.33 x 10^5	0.239	\$1.03 x 10^5
Total present value			\$9.73 x 10^5

Capital recovery factor (20 years at 10%) = 0.117
Levelized annual cost of capital equipment = \$1.14 x 10^5

II. Labor cost

Truck drivers and equipment operators--1973 wages assumed to be \$8.97/hr*

A. Labor cost (trucks) = \$8.97/hr x 4.5 drivers x 24 hr/day x 260 days/year
= \$251,878/year

B. Labor cost (loader) = \$8.97/hr x 1 operator x 24 hr/day x 260 days/year
= \$55,973/year

C. Labor cost (bulldozer) = \$8.97/hr x 0.33 operators x 8 hr/day x 260 days/year
= \$6,157/year

D. Total labor cost = \$314,008/year

TABLE 90 (CONTINUED)

III. Supervision

Assumed to be 25% of labor cost***
Supervision = 0.25 x \$314,008/year = \$78,502/year

IV. Maintenance

Assumed to equal 6% of nominal cost of equipment
Maintenance = 0.06/year x \$4.33 x 10^5 = \$25,980/year

V. Insurance and taxes

Assumed to equal 2%/year of nominal capital cost (land and equipment)
Insurance and taxes = 0.02/year x (\$1.69 x 10^6 + \$4.33 x 10^5) = \$4.2 x 10^4/year (Case B)

VI. Energy and power

A. Trucks

85-ton trucks assumed to consume 15 gal/hr†
Diesel fuel assumed to cost \$0.50/gal (1973 dollars)‡

Energy plus power (trucks) = 4.5 trucks x 24 hr/day x 260 days/year x 15 gal/hr x \$0.50/gal.
= \$210,600/year

B. Loader

7.65 m^3 (10 cu yd) front loader is assumed to consume 18 gal/hr§

Energy plus power (loader) = 1 loader x 24 hr/day x 260 days/year x 18 gal/hr x \$0.50/gal
= \$56,160/year

C. Bulldozer

1 bulldozer is assumed to use 60 gal/8 hr*

Energy plus power (bulldozer) = 0.33 bulldozers x 60 gal/8 hr x 260 days/year x \$0.50/gal.
= \$2,574/year

TABLE 90 (CONCLUDED)

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* U.S. Environmental Protection Agency. Assessment of industrial hazardous waste practices, inorganic chemical industry. Contract No. 68-01-2246, 1975. p. 7-33. Estimates were obtained for the 10-ton truck and the "0.6 power factor" was used to estimate the cost of the 85-ton truck.

† U.S. Environmental Protection Agency. Assessment of industrial hazardous waste practices, inorganic chemical industry. Contract No. 68-01-2246, 1975, p. 7-33. Estimates were obtained for the 10-ton truck and the "0.6 power factor" was used to estimate the cost of the 85-ton truck.

‡ MRI communication with Caterpillar Corporation, Kansas City, Missouri, June 3, 1976.

§ Using M+S Equipment Cost Index, Chemical Engineering, May 24, 1976 and February 18, 1974.

** U.S. Environmental Protection Agency. Assessment of industrial hazardous waste practices, inorganic chemical industry. Contract No. 68-01-2246, 1975, p. 7-33. Estimates were obtained for the 10-ton truck and the "0.6 power factor" was used to estimate the cost of the 85-ton truck.

TABLE 91

DISPOSAL COSTS--POTENTIALLY HAZARDOUS WASTE FROM OPEN-PIT URANIUM MINING--MINE BACKFILLING, TECHNOLOGY LEVEL I (EXPRESSED AS ANNUAL 1973 (Q4) DOLLARS)

	Case "A"*	Case "B"†	Case "C"‡	Case "D"§
Capital				
Land	$\$2.36 \times 10^5$	$\$1.98 \times 10^5$	$\$1.87 \times 10^5$	-
Equipment	$\$1.14 \times 10^5$	$\$1.14 \times 10^5$	$\$1.14 \times 10^5$	$\$1.14 \times 10^5$
Labor	$\$3.14 \times 10^5$	$\$3.14 \times 10^5$	$\$3.14 \times 10^5$	$\$3.14 \times 10^5$
Supervision	$\$7.85 \times 10^4$	$\$7.85 \times 10^4$	$\$7.85 \times 10^4$	$\$7.85 \times 10^4$
Maintenance	$\$2.60 \times 10^4$	$\$2.60 \times 10^4$	$\$2.60 \times 10^4$	$\$2.60 \times 10^4$
Insurance plus taxes	$\$4.90 \times 10^4$	$\$4.20 \times 10^4$	$\$4.20 \times 10^4$	$\$8.66 \times 10^3$
Energy and power	$\$2.70 \times 10^5$	$\$2.70 \times 10^5$	$\$2.70 \times 10^5$	$\$2.70 \times 10^5$
Total cost of mine backfilling	$\$1.09 \times 10^6$	$\$1.01 \times 10^6$	$\$1.03 \times 10^6$	$\$8.10 \times 10^5$
Ratio of potentially hazardous waste rock to overburden	0.6%	0.6%	0.6%	0.6%
Cost of hazardous waste disposal	$\$6.5 \times 10^3$	$\$6.2 \times 10^3$	$\$6.2 \times 10^3$	$\$4.9 \times 10^3$
Metric tons of potentially hazardous waste to mine backfill per year	30.7×10^3	30.7×10^3	30.7×10^3	30.7×10^3
Cost per metric ton of potentially hazardous waste in mine backfill	$0.21	$0.20	$0.20	$0.16
Total cost if entire industry adopted ($\$10^6$/yr)	$0.30			
Total cost as percent of value added	0.02%			0.01%

* Case "A" assumes all land required is purchased in year 1 and has no resale value after 20 years.

† Case "B" assumes all land required is purchased in year 1 and has the same value when resold after 20 years.

‡ Case "C" assumes 32.7 ha are bought per year for 5 years and then held for 15 years, and have 0 value after 20 years.

§ Case "D" assumes land has no rent or cost to the company because it is owned by the firm for future mining (i.e., the land has no opportunity cost).

TABLE 92

MAJOR COST ASSUMPTIONS: TECHNOLOGY LEVELS II AND III, URANIUM OPEN-PIT MINE, SURFACE WASTE DUMP

Note: In technology Levels II and III, the potentially hazardous waste rock is segregated from the overburden.

I. Land requirement--representative open-pit mine

425,700 MT (469,300 tons) of waste rock produced per year by representative plant
395,000 MT (435,400 tons) of waste rock per year are deposited in surface dump
2.01×10^5 cu m deposited at dump per year at bulk density of 1.96 MT/cu m (1.5 MT/cu yard)
After 20 years, 4.03×10^6 cu m will be deposited
Dump assumed to be 7.62 m deep (25 ft)
Therefore, 5.28×10^5 sq m or 52.8 hectares (131 acres) are required

II. Capital cost

A. Land cost = 52.8 hectares (130.5 acres) at $12,350/hectare = $$6.52 \times 10^5$
Levelized annual cost (assuming land is purchased in year 1 and has no value after 20 years) = $76,725
B. Bulldozer
Levelized annual cost of one bulldozer = $14,716*
Bulldozer assumed to be required 3% of time
Levelized annual cost of 3% of one bulldozer = $441/year

III. Materials and revegetation costs

All revegetation (i.e., topsoil cover, seeding and fertilizer) assumed to be done when surface dump becomes inactive (i.e., after 20 years)
4 in. soil and seeding (includes materials and labor) = $742 - $1,484/hectare ($300 - $600/acre)[39]
12 in. soil cover and seeding (includes materials and labor) = $1,854 - $4,325/hectare ($750 - $1,750/acre)[39]

	4 in. soil		12 in. soil	
	Low	High	Low	High
Total revegetation cost (52.8 ha, 131 acres)	$39,300	$78,600	$98,250	$229,250
Present value (20 years at 10%)	6,445	12,890	16,113	37,597
Levelized annual cost†	754	1,508	1,885	4,399

TABLE 92 (CONCLUDED)

IV. Labor cost

Bulldozer operator = \$18,658* x 0.03 = \$560/year

V. Supervision

Assumed to be 25% of labor cost or \$140/year

VI. Maintenance

Bulldozer maintenance = \$3,360/year* x 0.03 = \$100/year
Maintenance on revegetated land is minimal

VII. Energy and power

Bulldozer energy and power costs = \$7,800/year* x 0.03 = \$234/year

VIII. Insurance and taxes

Assumed to equal 2% of nominal capital cost (land and equipment)[39/]
Insurances and taxes = 0.02 x (\$6.52 x 10^5 + \$1.7 x 10^3) = \$1.3 x 10^4

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* See Table 89, p. 223.
† Capital recovery factor of 0.117.

The disposal costs of potentially hazardous waste from open-pit uranium mines in surface waste dumps are presented in Table 93. A low and high cost estimate is presented for 4 and 12 in. of top soil covering. Fertilizer costs are not included. Fertilizer costs would not alter the cost per metric ton estimate because these costs are relatively small.[33] The table indicates that the different cost estimates of revegetation have little impact on the disposal cost per metric ton. Differences in the depth of the top soil only change the cost per metric ton by $0.01.

Added Costs for Protective Cover on Active Surface Waste Dumps--Level I/II Technology. The preceding cost analysis was based on a 20-year life for a dump, and with stabilization being affected at the end of the 20-year period when the dump becomes inactive. If overburden is available for periodic emplacement on potentially hazardous waste rock, no added expense should be incurred. If, however, the cover material must be generated on-site, added costs will be incurred for bulldozing adjacent soil materials and emplacing them periodically over exposed potentially hazardous wastes.

A variety of schedules and dump construction procedures may in principal be selected. The following cost analysis is based on the assumption that a protective layer 0.3-m (1-ft) thick is spread over an annual accumulation of waste rock. It is further assumed that one-fourth of the area needed for a 20-year period is active at any one time. The assumptions are summarized below.

Total area required for 20 years - 52.8 hectares (131 acres)

Area active in a given year - 13.2 hectares (32.7 acres)

Waste rock per year - 395,000 MT

Waste rock volume per year - 201,000 m^3

Waste rock depth per year - 1.52 m (5 ft)

Cover depth per year - 0.305 m (1 ft)

Cost of annual cover per unit area - $3,950/hectare ($1,600/acre)

Since the costs will be incurred annually, no levelization is necessary. The added costs are thus calculated as follows.

TABLE 93

DISPOSAL COSTS - POTENTIALLY HAZARDOUS WASTES FROM OPEN-PIT URANIUM MINING - SURFACE WASTE DUMP, TECHNOLOGY LEVELS II AND III (EXPRESSED IN EQUIVALENT ANNUAL 1973 Q4 DOLLARS)

	Assuming 4 in. of Topsoil on Dump		Assuming 12 in. of Topsoil on Dump	
	Low Estimate	High Estimate	Low Estimate	High Estimate
Capital (land and equipment)	$76,725[a] (63,773)[b]	$76,725	$76,725	$76,725
Materials and Revegetation	754	1,508	1,885	4,399
Labor	560	560	560	560
Supervision	140	140	140	140
Maintenance	100	100	100	100
Insurance and Taxes	13,000	13,000	13,000	13,000
Energy and Power	234	234	234	234
Total Cost of Surface Dump (Technologies II and III)	$91,513 (78,561)	$92,267	$92,644	$95,158
Metric tons of potentially hazardous waste to surface dump each year	395×10^3	395×10^3	395×10^3	395×10^3
Cost per metric ton of potentially hazardous waste	$0.23 (0.20)	$0.23	$0.24	$0.24
Total cost if entire industry adopted ($\$10^6$/yr)	$0.38	-	-	$0.45
Total cost as a percent of value added	0.23%	-	-	0.28%

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[a] This value assumes the land is all purchased in the first year and has no resale value after 20 years.

[b] This estimate, in parenthesis, assumes the land is purchased in the first year and retains 100% of its value when resold after 20 years.

Total costs for annual addition of cover = 13.2 hectares by $3,950/ hectare = $52,140

Cost per metric ton of potentially hazardous waste = $52,140 ÷ 395,000 MT = $0.13/MT

The costs will decrease in direct proportion to a decrease in the fraction of the total dump which is active at a given time and will increase in proportion to a shortening of the period between applications of cover. Thus, the cost will be $0.26/MT if one-fourth of the dump is active in a given interval and the surface is covered semiannually. Semiannual coverage of one-eighth of the total dump area will cost $0.13/MT.

The added cost of annual coverage of one-fourth of the total dump area will increase Level II/III costs estimated in Table 93 by about 55 percent and by about 57 percent of Level I costs. The total cost, if the entire industry used this technology would vary from $0.38 to $0.45 million per year and this would amount to less than 0.5 percent of the value added in mining.

Disposal Costs--Underground Uranium Mining, Surface Waste Dump.

The rationale and assumptions used to develop a representative model for underground mining operations are discussed on page 175 under "Mass Balances of Materials." The mass balance data for a representative model and for waste treatment and disposal practices of underground uranium mining operations are given in Figures 25 (p. 187), 30 (p. 210), and 31 (p. 212).

Level I Technology. The disposal of potentially hazardous waste from underground uranium mining is quite similar to open-pit uranium mining disposal techniques. Potentially hazardous waste rock is deposited on a surface dump, contoured (under Level I technology), and natural vegetation is allowed to occur. Major assumptions used in the calculation of these costs are presented in Table 94. Many of the assumptions used in the open-pit mine waste disposal calculations are also used in the analysis of underground mine waste disposal costs.

The results of the analysis of Level I technology are presented in Table 95. The cost per metric ton of potentially hazardous waste treated range from $0.25 to $0.04, depending on how land is acquired and valued. Total cost if the entire industry adopted the technology level would not exceed $50,000/year. The total cost would be less than 0.05 percent of the value added.

TABLE 94

MAJOR COST ASSUMPTIONS--UNDERGROUND URANIUM MINING SURFACE WASTE DUMP--TECHNOLOGY LEVEL I

I. Land requirements

51,000 MT/year (56,200 tons/yr) of potentially hazardous waste rock are deposited on dump in the representative plant
26,020 cu m/year (34,000 cu yd/yr) are deposited assuming a bulk density of 1.96 MT/cu m
Over 20 years, 520,408 cu m are deposited
The dump is assumed to be 7.62 m deep (25 ft)
Therefore, land requirements equal 68,295 sq m or 6.8 hectares (16.8 acres)

II. Capital costs

A. Land purchase price at $12,350/hectares ($5,000/acre) = $83,980

B. Other capital costs
Note: The land area required for the dump from the underground mine is approximately 1% of the open-pit dump. Therefore, costs will be scaled down from the open-pit calculations presented earlier.

Bulldozer: $14,716/year[40] x 0.01 = $147/year

III. Labor cost

$18,658[40] x 0.01 = $186/year

IV. Supervision

$4,664[40] x 0.01 = $47/year

V. Maintenance

$3,360[40] x 0.01 = $34/year

VI. Insurance and taxes

Assumed to equal 2% of nominal capital cost

VII. Energy and power

$7,800[40] x 0.01 = $78/year

TABLE 95

DISPOSAL COSTS, POTENTIALLY HAZARDOUS WASTE FROM UNDERGROUND URANIUM MINING - SURFACE WASTE DUMP, LEVEL I TECHNOLOGY (EXPRESSED IN ANNUAL EQUIVALENT 1973 DOLLARS)

	Case "A"*	Case "B"†	Case "C"‡	Case "D"§
Capital (land and equipment)	$ 9,973	$ 8,361	$ 4,364	$ 147
Labor	186	186	186	186
Supervision	47	47	47	47
Maintenance	34	34	34	34
Materials	None	None	None	None
Insurance and taxes	1,705	1,705	1,705	1,705
Energy and power	78	78	78	78
Total cost	$12,023	$10,411	$ 6,414	$ 2,197
Metric tons of potentially hazardous waste to surface dump per year	51,000	51,000	51,000	51,000
Cost per metric ton of potentially hazardous waste in surface dump per year	$ 0.25	$ 0.20	$ 0.13	$ 0.04
Total cost if entire industry adopted ($$10^6$/yr)	$ 0.05	-	-	$ 0.008
Total cost as a percent of value added	0.03%	-	-	0%

* Assumes all land purchased in year 1 and has 0 value after 20 years.

† Assumes all land is purchased in year 1 and is resold at the same value after 20 years.

‡ Assumes land is purchased yearly (247 acres) and has 0 value after 20 years.

§ Assumes land would be purchased even if no waste disposal was required, and therefore, land costs do not enter cost calculation.

Levels II and III Technology. Disposal of potentially hazardous wastes using Level II technology is the same as Level III technology. The surface dump is contoured, covered with top soil, seeded, and fertilized, if necessary. In New Mexico, where most underground uranium mines are located, rock covering may be less costly than top soil and seeding. Costs for both Levels II and III technology are presented in Table 96. Costs are presented for four alternative types and depth of dump covering (i.e., 4 and 12 in. of top soil and seeding, 6 and 12 in. of rock covering). Costs range from $0.31 to $0.06/MT of potentially hazardous waste, depending on how land costs are handled. The cost to the industry as a whole would be $0.038 to $0.088 million per year, assuming all underground uranium mines used these technology levels. This cost represents less than 0.5 percent of the value added in uranium mining.

Waste Treatment Costs - Wastes from Acid and Alkaline Leaching Uranium Concentrators. Disposal of potentially hazardous wastes occurring in the concentration of uranium can be accomplished in tailings ponds. Level I technology requires the construction of an earthdike tailings pond which allows zero discharge of wastewater. The tailings pond is assumed to be within 914 m (3,000 ft) of the model plant. Natural vegetation is allowed on the dam and tailings beaches.

The model plant used in the following analysis is taken from a recent study by M. B. Sears.[34] Much of the plant description and capital cost data used below are drawn from the Sears study. The rationale used in selecting the model plants is discussed on page 213. The mass balance data for the model plants are given in Figure 24 (p. 181) and Figure 25 (p. 187).

The tailings pond of the model plant is sited within 914 m (3,000 ft) of the mill. It is near the upper reaches of a gently sloping natural drainage area and at least 61 m (200 ft) from any surface stream or permeable formation. A starter dam is constructed of native soil across the lower end of the site. The remainder of the dam is built from the tailings. The tailings are hydraulically classified by hydroclones or settling. The maximum height of the dam is assumed to be 30 m (100 ft), including 1.5 m (5 ft) of freeboard.[35] A minimum 0.8 hectares (2 acres) beach area of dry tailings sand is maintained along the dam to protect against wave action and dam instability due to liquefaction. The face of the dam contains exposed tailings. The tailings pond is designed to keep the maximum area of tailings wet. A 20 percent contingency of land is included for abnormal weather conditions. Under Level I technology, the beaches and tailings dam are covered with natural vegetation.

The capital costs of the tailings pond (Level I technology) are illustrated in Table 97. Both acid and alkaline leach processes are presented for Wyoming and New Mexico locations. Land area requirements vary because of the different leach processes and different average evaporation rates in Wyoming and New Mexico. The costs of the tailings pump and pipeline, earth dam, and cyclone are also presented. Table 98 presents similar information on operational costs of the pond.

TABLE 96

DISPOSAL COSTS, UNDERGROUND URANIUM MINING LEVELS II AND III TECHNOLOGY, SURFACE WASTE DUMP (EXPRESSED IN EQUIVALENT ANNUAL 1973 DOLLARS)

	Case "A"*	Case "B"†	Case "C"‡	Case "D"§
Capital	$ 9,973	$ 8,361	$ 4,364	$ 147
Labor	186	186	186	186
Supervision	47	47	47	47
Maintenance	34	34	34	34
Insurance and taxes	1,705	1,705	1,705	1,705
Energy and power	78	78	78	78
Revegetation				
10-cm (4-in.) topsoil and seeding at $450/acre[19]	884	884	884	884
30.5-cm (12-in. top soil and seeding at $1,250/acre[19]	2,457	2,457	2,457	2,457
15-cm (6-in.) rock covering at $1,000/acre[19]	1,966	1,966	1,966	1,966
30.5-cm (12-in.) rock covering at $2,000/acre[19]	3,931	3,931	3,931	3,931
Total				
10-cm topsoil	$12,907	$11,295	$ 7,298	$ 3,081
30.5-cm topsoil	14,480	12,868	8,871	4,654
15-cm rock	13,989	12,377	8,380	4,163
30.5-cm rock	15,954	14,342	10,345	6,128
Metric tons of potentially hazardous waste to dump per year	51,000	51,000	51,000	51,000
Cost per metric ton of potentially hazardous waste in surface dump per year				
10-cm topsoil	$ 0.25	$ 0.22	$ 0.14	$ 0.06
30.5-cm topsoil	0.28	0.25	0.17	0.09
15-cm rock	0.27	0.24	0.16	0.08
30.5-cm rock	0.31	0.28	0.20	0.12
Total cost if industry adopted ($$10^6$/yr)	0.088	-	-	0.038
Total cost as a percent of value added	0.05%	-	-	0.02%

* Case "A" assumes all land purchased in year 1 and has 0 value after 20 years.

† Case "B" assumes all land is purchased in year 1 and is resold at the same value after 20 years.

‡ Case "C" assumes land is purchased yearly (247 acres) and has 0 value after 20 years.

§ Case "D" assumes land would be purchased even if no waste disposal was required, and therefore, land costs do not enter cost calculation.

TABLE 97

CAPITAL COST ASSUMPTIONS FOR URANIUM TAILINGS PONDS (LEVEL I TECHNOLOGY) IN WYOMING AND NEW MEXICO (EXPRESSED IN 1973 DOLLARS)

	Wyoming		New Mexico	
	Acid leach ($)	Alkaline leach ($)	Acid leach ($)	Alkaline leach ($)
Land area in hectares (acres)19/	70 (173)	44 (109)	39 (96)	24 (60)
Land cost at $12,350/hectare ($5,000/acre)	870,000	540,000	480,000	300,000
Tailings pump and pipeline19/	250,000	156,000	138,000	134,000
Earth dam19/	63,000	40,000	35,000	33,000
Cyclone installation19/	58,000	58,000	58,000	58,000
Total capital cost (Case "A")*	1,241,000	794,000	711,000	525,000
Levelized annual cost (A)†	145,000	93,000	83,000	61,000
Total capital cost (Case "B")‡	1,098,000	705,000	632,000	476,000
Levelized annual cost (B)†	128,000	83,000	74,000	56,000

* Case "A" assumes the land has 0 value after 20 years of analysis.

† Assuming a 10% discount rate and 20 years of analysis. The capital recovery factor is 0.117.

‡ Case "B" assumes the land is resold after 20 years at the same value as the purchase price.

TABLE 98

OPERATING COST ASSUMPTIONS - TAILINGS POND (LEVEL I TECHNOLOGY)

I. Labor cost[40/]

One operator required on cyclone separators for 5 months, 8 hr/day at $8.97/hr
Labor = $7,176/year

II. Supervision

Assumed to be 25% of labor cost[40/]
Supervision = $1,794/year

III. Maintenance

Assumed to be equal to 6% of nominal cost of equipment per year[40/]

	Wyoming		New Mexico	
	Acid ($)	Alkaline ($)	Acid ($)	Alkaline ($)
Cost of tailings pump, pipeline, and cyclone	308,000	214,000	196,000	192,000
Total maintenance cost per year	18,000	13,000	12,000	12,000

IV. Insurance and taxes

Assumed to equal 2%/year of total nominal capital cost (land and equipment[40/]
Insurance and taxes = 0.02/year by ($1.2 x 10^6) = $24,000 (Wyoming, acid-leach, Case "A")

V. Energy and Power

A. Tailings pump assumed to consume electricity with a value of approximately $500/year[40/]
B. Cyclone separator mounted on truck--only fuel required is for truck as it travels around the tailings pond; assumed to be less than $100/year

The inputs from Tables 97 and 98 were combined to obtain the results shown in Tables 99 and 100. Table 99 presents the total cost of building and operating the model tailings pond in Wyoming. Costs are shown for both the acid and alkaline leach processes. The table shows that the tailings pond requirements of the alkaline leaching process are approximately $0.10 cheaper per metric ton of handled waste than the tailings pond associated with the acid leaching process in Wyoming.

Table 100 shows the cost of disposing of potentially hazardous wastes in New Mexico. The costs per metric ton of potentially hazardous wastes are significantly lower (using Level I technology) in New Mexico than in Wyoming. This cost reduction occurs because less land area is required in New Mexico. Tables 99 and 100 also present the cost to the industry as a whole, and their relative magnitude as a percent of value added.

Disposal Costs--Uranium Tailings Pond, Levels II and III Technology. Uranium tailings pond disposal costs are similar for all three levels of technology. The only difference is that revegetation of the tailings dam surface and dried exposed tailings (tailings beaches) is carried out in Levels II and III technology. The surface area of the dam and the amount of exposed dry tailings depends on the location of the pond and the type of concentrator used. Table 101 presents the assumed areas used in the analysis. The analysis further assumes that the area on the face of the dam is covered with soil and seeded in the 1st year of the analysis. The dry beach area is assumed to be covered in year 16. (Therefore, it is discounted by 0.239.) Using the land areas in Table 101, the major cost components were derived (see Table 97, p. 239.

The results for Wyoming and New Mexico are presented in Tables 99 and 100, respectively. In Wyoming estimates are presented for 4 and 12 in. of topsoil cover. In New Mexico, rock covering of 6 and 12 in. is also estimated.

TABLE 99

DISPOSAL COSTS, POTENTIALLY HAZARDOUS WASTE FROM OPEN-PIT URANIUM MINING--ACID LEACH AND ALKALINE LEACH CONCENTRATORS--LEVEL I TECHNOLOGY IN WYOMING (EXPRESSED IN EQUIVALENT ANNUAL 1973 DOLLARS)

	Acid leach		Alkaline leach	
	Case "A"*	Case "B"†	Case "A"*	Case "B"†
Capital (levelized cost)	$145,000	$128,000	$ 93,000	$ 83,000
Labor	7,176	7,176	7,176	7,176
Supervision	1,794	1,794	1,794	1,794
Maintenance	18,000	18,000	13,000	13,000
Insurance and taxes	24,000	22,000	16,000	14,570
Energy and power	600	600	600	600
Total cost	$196,570	$177,570	$131,570	$119,570
Metric tons of potentially hazardous tailings deposited in pond annually	662,200	662,200	662,200	662,200
Cost per metric ton of potentially hazardous waste	$ 0.30	$ 0.27	$ 0.20	$ 0.18
Total cost if entire industry adopted ($$10^6$/yr)	$ 0.84	-	-	$ 0.50
Total cost as a percent of value added	0.5%	-	-	0.3%

* Case "A" assumes the land is all purchased in the initial year and has no value after 20 years.

† Case "B" assumes the land is all purchased in the initial year but has the same dollar value after 20 years of use.

TABLE 100

DISPOSAL COSTS, POTENTIALLY HAZARDOUS WASTE FROM OPEN-PIT URANIUM MINING--ACID LEACH AND ALKALINE LEACH CONCENTRATORS, LEVEL I TECHNOLOGY IN NEW MEXICO (IN ANNUAL 1973 DOLLARS)

	Acid leach		Alkaline leach	
	Case "A"*	Case "B"†	Case "A"*	Case "B"†
Capital (levelized cost)	$ 83,000	$ 74,000	$ 61,000	$ 56,000
Labor	7,176	7,176	7,176	7,176
Supervision	1,794	1,794	1,794	1,794
Maintenance	12,000	12,000	12,000	12,000
Insurance and taxes	14,000	13,000	11,000	10,000
Energy and power	600	600	600	600
Total cost	$118,570	$108,570	$ 93,570	$ 87,570
Metric tons of potentially hazardous tailings deposited in pond annually	662,200	662,200	662,200	662,200
Cost per metric ton of potentially hazardous waste	$ 0.18	$ 0.16	$ 0.14	$ 0.13
Total cost if entire industry adopted ($$10^6$/yr)	$ 0.51	-	-	$ 0.36
Total cost as a percent of value added	0.32%	-	-	0.2%

* Case "A" assumes the land is all purchased in the initial year and has no value after 20 years.

† Case "B" assumes the land is all purchased in the initial year but has the same dollar value after 20 years of use.

TABLE 101

LAND AREA OF URANIUM TAILINGS POND DAM AND EXPOSED BEACHES (EXPRESSED IN HECTARES (ACRES))

	Average area of dry beach near end of 20-year life of mill	Area of exposed tailings on face of dam
New Mexico		
Acid leach	14.6 (36)	4.9 (12)
Alkaline leach	26.7 (66)	4.9 (12)
Wyoming		
Acid leach	0.8 (2)	4.0 (10)
Alkaline leach	10.1 (25)	4.9 (12)

Source: Reference 19.

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SECTION VI

MISCELLANEOUS METAL ORES (SIC 1099)

Various miscellaneous metals are produced in the United States in small quantities by a few mines. The miscellaneous metals covered in this chapter include antimony, beryllium, platinum-group metals (platinum, palladium, rhodium, iridium, ruthenium, and osmium), rare earth, tin, titanium, and zirconium.

In 1974 there were nine mines producing these metals (Figure 32). Total production was estimated at 503,666 MT (554,000 tons). The greatest concentration of tonnage of primary metal products was in the production of zirconium and titanium ores from mines in Florida and New Jersey (Table 102).

Employment, according to the Engineering & Mining Journal, Mining Enforcement and Safety Administration, unpublished company data and Midwest Research Institute analyses, was about 1,328 workers. Most of these mines are small, employing fewer than 100 employees (Tables 103 and 104). Mine production data, where available in published commodity reports, for each of the metals are presented in Tables 105 through 107. Production data for titanium, zirconium, and beryllium were obtained for 1974 via questionnaires and visits. These data are reported in later sections which treat these metals individually.

Antimony

Industry Characterization.

History of the Industry. Antimony is one of the oldest metals continuously used by man. The natural sulfide of antimony was known and used by women in Biblical times as medicine and as a cosmetic for eyebrow painting. By the 15th century, antimony was used in alloys for printer's type, mirrors, and bells.

Precipitation of metal from the sulfide by iron is described by Ercker in the 17th century, and in the 18th century roast-reduction procedures came into use. The early 1830's marked the introduction of the reverberatory furnace for smelting antimony; 1844, the French volatilization process, and 1896, the first appearance of electrolytic antimony on the market.

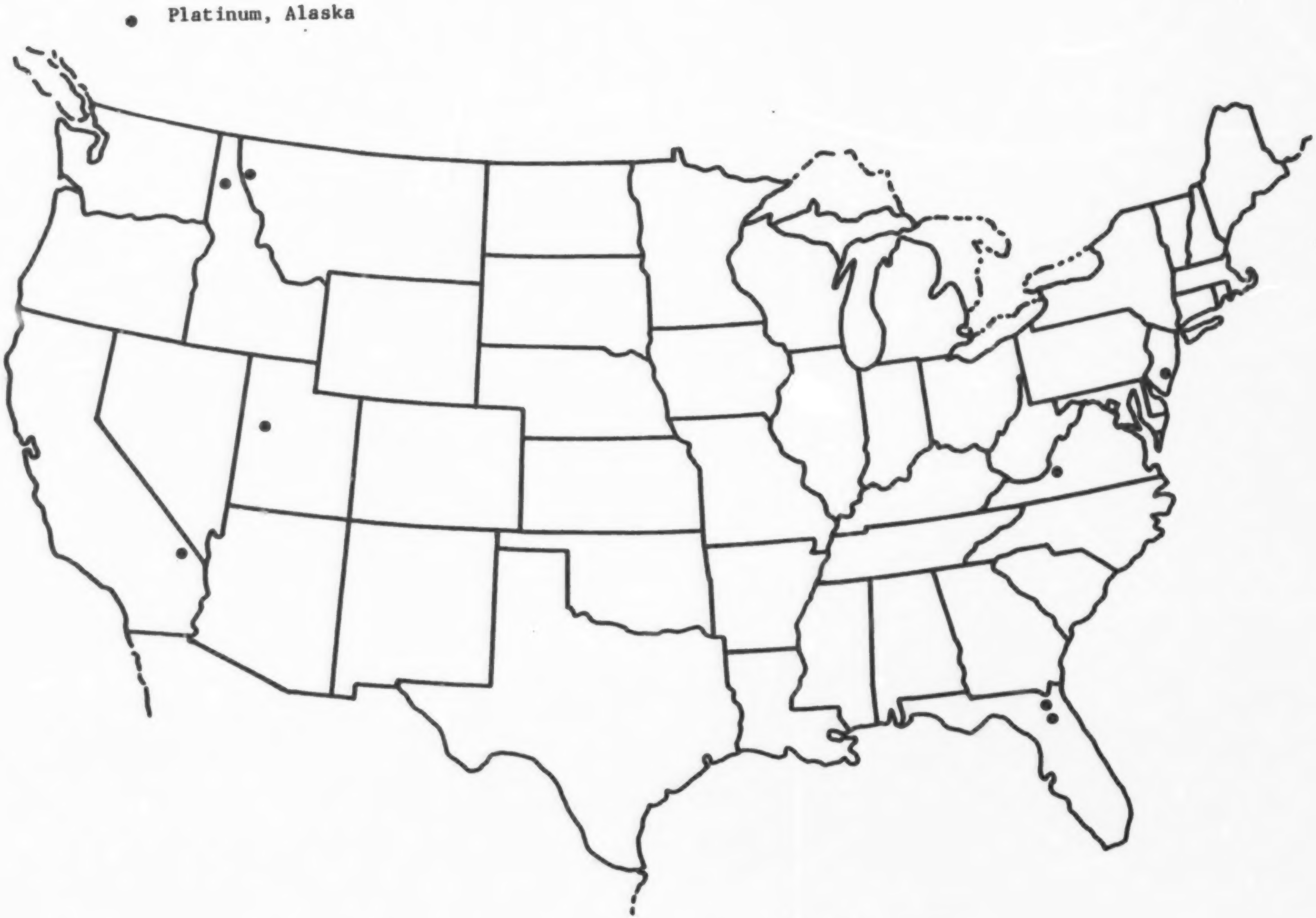

● Platinum, Alaska

Figure 32. Location of miscellaneous ores (SIC 1099).

TABLE 102

U.S. PRODUCTION OF MISCELLANEOUS METALS - 1974 (RECOVERED METALS AND CONCENTRATES)*

State	Metric tons	EPA regions	Metric tons
Alaska	NA	Region I	--
California	NA	Region II	113,000
Florida	388,000	Region III	--
Idaho	907	Region IV	388,000
Montana	150	Region V	--
New Jersey	113,000	Region VI	--
Utah	839	Region VII	--
U.S. total	502,896	Region VIII	989
		Region IX	NA
		Region X	907

Source: References 1 and 2.

NA = Not available.

* Includes antimony, beryllium, platinum, titanium, rare earth ores, tin, and zirconium.

TABLE 103

EMPLOYMENT IN ACTIVE MISCELLANEOUS MINING OPERATIONS (1974)

State	Employment	EPA regions	Employment
Alaska	37	Region I	--
California	93	Region II	80
Florida	300	Region III	41
Idaho	700	Region IV	300
Montana	14	Region V	--
New Jersey	80	Region VI	--
Utah	63	Region VII	--
Virginia	41	Region VIII	77
U.S. total	1,328	Region IX	93
		Region X	737

Source: References 1, 2, and 3.

TABLE 104

NUMBER, LOCATION, AND SIZE OF MISCELLANEOUS MINING, AND MILLING OPERATIONS (SIC 1099)
(1974)

State	Employees								Total operations	Total No. of mines	No. of mills
	1-19	20-49	50-99	100-249	250-499	500-999	1,000-2,499	2,500+			
Alaska		1							1	1	1
California			1						1	1	1
Florida				2					2	2	2
Idaho						1			1	1	1
Montana	1								1	1	1
New Jersey			1						1	1	1
Utah	1								1	1	1
Virginia		1							1	1	1
U.S. total	2	2	2	2		1			9	9	9
EPA regions											
Region I											
Region II			1						1	1	1
Region III		1							1	1	1
Region IV				2					2	2	2
Region V											
Region VI											
Region VII											
Region VIII	2								2	2	2
Region IX				1					1	1	1
Region X		1				1			2	2	2

Source: References 1, 2, and 3.

TABLE 105

WORLD ANTIMONY PRODUCTION, 1969-1973
(METRIC TONS)*

Year	United States	Rest of world	Total
1969	850	65,682	66,532
1970	1,025	65,337	66,362
1971	929	65,516	66,445
1972	443	67,628	68,071
1973	589	70,761	71,350

Source: References 4, and 6.

* Data published in tons and calculated to metric tons.

TABLE 106

WORLD PRODUCTION OF PLATINUM-GROUP METALS, 1968-1973
(KILOGRAMS)*

Year	United States	Rest of world	Total
1968	467	105,099	105,566
1969	684	106,032	106,716
1970	529	131,319	131,848
1971	560	126,467	127,027
1972	529	142,952	143,481
1973	622	160,618	161,147

Source: References 5 and 6.

* Data published in thousand troy ounces and calculated to kilograms.

TABLE 107

WORLD PRODUCTION OF PLATINUM, 1968-1972
(KILOGRAMS)*

Year	United States	Rest of world	Total
1968	218	44,851	45,069
1969	342	45,784	46,126
1970	249	61,149	61,398
1971	249	56,360	56,609
1972	156	68,148	68,304

Source: Reference 5.

* Calculated from troy ounces to kilograms.

Prior to the 20th century, chemical and metallurgical qualities of antimony found relatively small use. After the turn of the century, consumption increased rapidly as the world industrial pace quickened and the low-cost, abundant supply of Chinese ore became available.

Expansion of the automobile industry and the use of storage batteries stimulated consumption in the 1930's and in turn escalated secondary recovery of antimony. The escalated demand of the war years again reactivated idle mines in the United States during World War II. Antimony oxide used in flameproofing compounds for military use became the leading end product used and necessitated rapid expansion of domestic oxide-smelter capacity. Demand for consumer products kept antimonial lead consumption relatively high until the post-Korean War adjustment in 1953 caused a decline in foreign production, while domestic production was essentially terminated. Beginning in 1962, increased consumption in the United States and other industrialized countries resulted in a world supply deficit. Today the United States provides about 10 percent of its domestic requirements while importing the balance largely from the Republic of South Africa, Mexico, and Bolivia.

Domestic Production and Capacities. Lead-silver ores of the Coeur d'Alene district of Idaho contributed the majority of the 589 MT of primary antimony mined in the United States in 1973. Sunshine Mining Company and U.S. Antimony Corporation were the only domestic producers of antimony in the United States. Tetrahedrite concentrates from Sunshine Mining Company, Hecla Mining Company, and Silver Dollar Mining Company were processed into cathode metal, 96 percent antimony, at the Sunshine Mining Company's electrolytic plant. Table 108 gives the U.S. mine production of antimony by year.

The U.S. production capacity is much higher than the current production. Seven hundred workers were engaged in domestic mine production in 1974. World antimony production for the period 1969 to 1973 is shown in Table 105. Demand for antimony is expected to increase at an annual rate of about 4 percent through 1983. Since domestic output already is far below demand, antimony oxide producers will have to increase their imports of finished oxide, crude oxide, and ore.

By-Products and Coproducts. Most of the antimony from domestic sources is a coproduct of mining, smelting, and refining of other metals and ores that contain relatively small quantities of antimony. Because of the close mineralogical association of antimony with lead, antimony is extracted at primary lead refineries.

TABLE 108

PRIMARY MINE PRODUCTION OF ANTIMONY AND ANTIMONY CONCENTRATES IN THE UNITED STATES BY YEAR

Year	Antimony (MT)	Antimony concentrates (MT)
1969	851	5,177
1970	1,025	6,060
1971	930	4,282
1972	444	1,879
1973	494	2,238

Source: References 4 and 6.

Waste Generation and Characterization.

Waste Generation. Table 109 contains the national production data for antimony mining and concentration operations in 1974 (i.e., the statistics shown for Idaho and Montana apply for antimony only). In 1974, the total ore mined was 198,000 MT (218,000 tons), and the total antimony products amounted to 2,666 MT (2,930 tons). The total mining waste, consisting of waste rock only, was 90,000 MT (99,208 tons), and the total concentrator waste was 189,000 MT (208,000 tons).

Table 110 shows the ratios of waste rock and of tailings to ore mined for antimony operations (all data shown for Idaho and Montana). In Idaho, the ratio of waste rock to ore mined in 1974 was 0.47, and the ratio of concentrator tailings to ore was 0.98. For Montana, the waste rock-to-ore ratio was 0.26 and the tailings-to-ore ratio was 0.63.

Tables 111, 112, and 113 show the 1974 and projected 1977 and 1983 potentially hazardous waste resulting from antimony mining and concentrating operations in Idaho and Montana.

Underground Mining Process.

Description of Typical Process. A flow sheet showing an antimony operation is presented in Figure 33. The two operating antimony mines employ underground mining methods similar to those used in the mining of lead-zinc. The scale of operation is quite different, however, between the two mines; 86 percent (907 MT) of the antimony is produced at one mine.

Conventional underground mining methods are used, with one mine employing stope and pillar methods and the other single bench, room, and pillar. One mine has two primary shafts and about 12 operating levels ranging from 945 to 1,646 m (3,100-5,400 ft) underground. The ore is broken by drilling and blasting operations. One mining method is called breast stoping with the stopes being about 30 m (98 ft) wide on each side of a pillar. The mine stopes are backfilled with sand and a cement cap. The second mine was opened in 1971 and is still under development.

At the mine visited, ore is hauled underground by rail to a hoist that lifts the ore to the surface, where it is sent to a crushing plant. Waste rock, which is removed from the mine in the same manner as the ore, is hauled by truck to a waste dump about 0.8 km (0.5 miles) from the mine.

Mass Balance of Materials. The mass balance data are shown in the table on Figure 33.

TABLE 109

PRODUCTION STATISTICS BY STATE AND EPA REGION--MISCELLANEOUS ORES, SIC 1099, FOR 1974 (METRIC)

			Wastes (10^3 TPY)					Products (TPY)				Ratio of dry		%	
State	EPA Region	Ore mined (10^3 TPY)	Waste rock	Overburden	Concentrator dry wastes	Concentrator wet waste (10^3 TPY)	Total dry waste	Primary concentrate	By-product concentrate	Miscel-laneous	Total products	Waste to ore	Waste to product	Total waste	Metal ore
New Jersey*	II	7,000	0	0	-	-	-	113,000	0	0	113,000	-	-	-	Ti-Zr
Florida*	IV	15,422	0	0	-	-	-	388,000	125,000	42,000	555,000	-	-	-	Ti-Zr
Montana		17	5	0	11	125	16	150	0	0	150	0.94	106		Sb
Utah		91	907	0	90	1,021	997	839	0	0	839	11	1,188		Be
	VIII	108	912	0	101	1,146	1,013	989	0	0	989	9.4	1,024	79.0	
California	IX	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	RE†
Idaho	X	181	85	0	178	2,178	263	907	1,360	249	2,516	1.45	105	21.0	Sb
National		22,711	997	0	279	3,324	1,276	502,896	125,036	42,976	671,505	0.06	1.9		

Source: References 1 and 2.

Note: - = No information.

NA = Not available.

* Approximately 96% of the material defined as ore is returned directly to the dredge site and is not counted as waste rock or concentrator wastes.

† RE = Rare earth group metals.

TABLE 110

RATIO OF TOTAL WASTE ROCK-OVERBURDEN-CONCENTRATOR WASTE TO ORE MINED, SIC 1099, MISCELLANEOUS ORES FOR 1974 (METRIC)*

State	EPA Region	Ore mined (10^3 TPY)	Total waste rock (10^3 TPY)	Ratio of total waste rock to ore	Overburden (10^3 TPY)	Ratio of total overburden to ore	Tailings (10^3 TPY)	Ratio of tailings to ore	Wet waste (10^3 TPY)	Ratio of wet waste to ore	Metal ore
New Jersey	II	7,000	0	0	0	0	-	-	-	-	Ti-Zr
Florida	IV	15,422	0	0	0	0	-	-	-	-	Ti-Zr
Montana		17	5	0.26	0	0	11	0.63	125	7.3	Sb
Utah		91	907	10.0	0	0	90	0.99	1,021	11.2	Be
	VIII	108	912	8.4	0	0	101	0.93	1,146	10.6	
California		NI	NI	NI	NI	NI	NI	NI	NI	-	RE
Idaho	X	181	85	0.47	0	0	178	0.98	2,178	12.0	Sb
National		22,711	997	0.04	0	0	279	0.01	3,324	0.15	

* Compiled from data in Table 108.

Note: - = No information.
NI = Not included.
RE = Rare earths.

TABLE 111

TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING MISCELLANEOUS ORES, SIC 1099, FOR 1974 (METRIC)

State	EPA Region	Dry waste weight (10^3 TPY)						Concentrator tailings (10^3 TPY)				Metal ore
		Total process waste	Potentially hazardous waste	Waste rock		Overburden		Dry weight		Wet weight		
				Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	
Montana	VIII	16	11	5	0	0	0	11	11	125	125	Sb
Utah	VIII	997	90	907	0	0	0	90	90	1,021	1,021	Be
Idaho	X	263	178	85	0	0	0	178	178	2,178	2,178	Sb
Total		1,276	279	997	0	0	0	279	279	3,324	3,324	Sb-Be

Source: References 2 and 8.

TABLE 112

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING MISCELLANEOUS ORES FOR SIC 1099, FOR 1977 (METRIC)

		Dry waste weight (10^3 TPY)						Concentrator tailings (10^3 TPY)				
				Waste rock		Overburden		Dry weight		Wet weight		
State	EPA Region	Total process waste	Potentially hazardous waste	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Metal ore
Montana	VIII	17	12	5	0	0	0	12	12	133	133	Sb
Utah	VIII	1,056	95	961	0	0	0	95	95	1,082	1,082	Be
Idaho	X	294	199	95	0	0	0	199	199	2,440	2,440	Sb
Total		1,367	306	1,061	0	0	0	306	306	3,655	3,655	

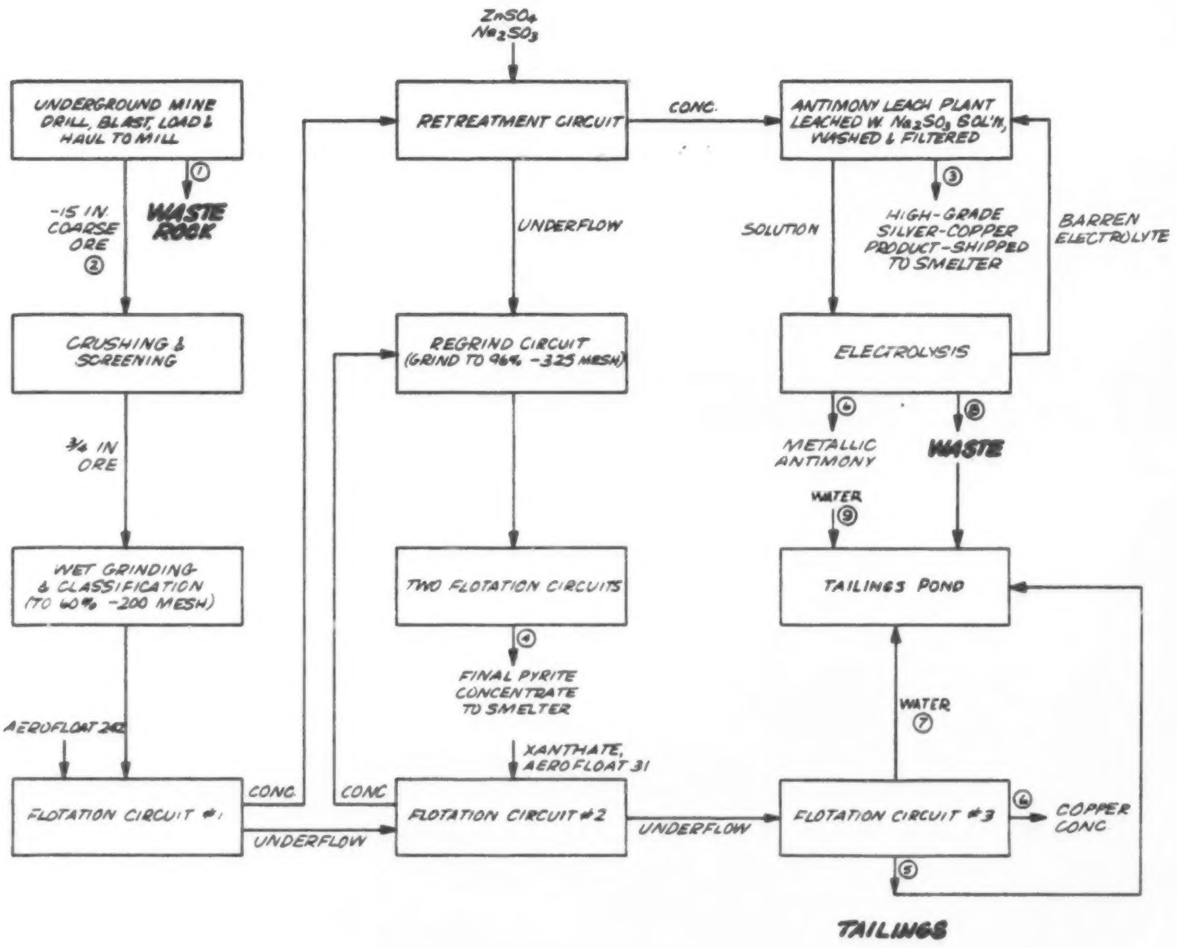
Source: References 2, 6, and 8.

TABLE 113

PROJECTED TOTAL AND POTENTIALLY HAZARDOUS WASTE FROM MINING AND CONCENTRATING MISCELLANEOUS ORES, SIC 1099, FOR 1983 (METRIC)

		Dry waste weight (10^3 TPY)						Concentrator tailings (10^3 TPY)				
				Waste rock		Overburden		Dry weight		Wet weight		
State	EPA Region	Total process waste	Potentially hazardous waste	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Total	Potentially hazardous	Metal ore
Montana	VIII	19	13	6	0	0	0	13	13	148	148	Sb
Utah	VIII	1,176	106	1,070	0	0	0	106	106	1,205	1,205	Be
Idaho	X	357	241	116	0	0	0	241	241	2,960	2,960	Sb
Total		1,552	360	1,192	0	0	0	360	360	4,315	4,315	

Source: References 2, 6, and 8.


ZnSO4
Na2SO3
UNDERGROUND MINE DRILL, BLAST, LOAD & HAUL TO MILL
①
WASTE ROCK
-15 IN. COARSE ORE
②
CRUSHING & SCREENING
¾ IN ORE
WET GRINDING & CLASSIFICATION (TO 60% -200 MESH)
AEROFLOAT 242
FLOTATION CIRCUIT #1
CONC.
UNDERFLOW
RETREATMENT CIRCUIT
CONC.
UNDERFLOW
REGRIND CIRCUIT (GRIND TO 96% -325 MESH)
TWO FLOTATION CIRCUITS
④
FINAL PYRITE CONCENTRATE TO SMELTER
XANTHATE, AEROFLOAT 31
CONC
FLOTATION CIRCUIT #2
UNDERFLOW
ANTIMONY LEACH PLANT LEACHED W. Na2SO3 SOL'N, WASHED & FILTERED
SOLUTION
③
HIGH-GRADE SILVER-COPPER PRODUCT-SHIPPED TO SMELTER
BARREN ELECTROLYTE
ELECTROLYSIS
⑥
METALLIC ANTIMONY
⑧
WASTE
WATER
⑨
TAILINGS POND
WATER
⑦
FLOTATION CIRCUIT #3
⑥
COPPER CONC
⑤
TAILINGS

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS	① WASTE ROCK	② CRUDE ORE	③ SILVER CONC.	④ PYRITE CONC.	⑤ FINAL TAILINGS	⑥ ANTIMONY	⑦ WATER FROM FLOTATION	⑧ ANTIMONY TAILINGS DRY	⑨ WATER FROM ANTIMONY CONCENTRATOR
Quantity, TPY	94,000	200,000	274	1,500	196,000	1,000	2,200,000	530	1,400
Metric TPY	85,000	181,000	249	1,360	178,000	907	2,000,000	480	1,260

FLOTATION ADDITIVES

NAME	LB/TON OF ORE	G/MT OF ORE
Aerofloat 242	0.015	7.50
Methyl Amyl Alcohol	0.07	35.00
Aerofloat 31	0.01	5.00
S-3477	0.02	10.00
Creosote Oil	0.01	5.00
Zinc Sulfide	0.30	150.02
Sodium Sulfite	0.40	199.96
Xanthate Z-11	0.005	2.50

Source: References 2 and 8.

Figure 33. Mining and concentrating of antimony.

Description of Individual Waste Streams. There is only one waste stream associated with the underground mining of antimony. This stream consists of waste rock that is disposed of in waste dumps. The waste rock consists mainly of siderite and quartz with trace amounts of the valuable minerals of the region.

Identification of Potentially Hazardous Materials. The waste rock from mining operations does not contain any materials that are considered potentially hazardous. The amount of total waste rock generated from antimony mining is about 30 percent of the total amount of materials handled.

Concentrator Processes.

Flotation. Flotation is used to recover antimony and other metals and prepare concentrate.

The number of circuits used in the flotation process varies depending upon the number of products to be recovered in conjunction with the antimony. A typical flow diagram for concentrating antimony using a three circuit-flotation process is shown in Figure 33.

Crushing Plant. Ore of 38 cm (15 in.) maximum size is hoisted from the mine to a coarse ore bin. From the bin it is fed to the crushing plant by an apron feeder. The ore is crushed to a size of less than 8 cm (3 in.) in the primary crusher and is passed to a secondary crusher which reduces the ore to 2.5 cm (1 in.).

Wet Grinding and Classification. The ore is wet-ground in three ball mills which are arranged in a closed-circuit with a classification unit. The ore is ground to 60 percent -200 mesh to free most of the valuable minerals from the waste rock. In normal operations, the three ball mills can grind between 1,000 and 1,100 MT (1,102 and 1,213 tons) of ore per day depending upon the hardness of the ore. The ore slurry is fed to the No. 1 flotation circuit.

No. 1 Flotation Circuit. This flotation circuit consists of Fagegren flotation cells grouped as two cleaners, four roughers, and four scavengers. Two reagents, Aerofloat 242 and methylisobutylcarbinol, are added in this circuit. These reagents promote the collection and froth flotation of the silver-bearing tetrahedrite and also the chalcopyrite. Approximately 96 percent of the silver-bearing tetrahedrite is recovered in this circuit. The concentrate from this circuit goes to the retreatment circuit. Underflow from the No. 1 flotation circuit is sent to the No. 2 flotation unit.

Retreatment Circuit. The retreatment circuit consists of eight flotation machines arranged as two cleaners, four roughers, and two conditioners. Chemical reagents ($ZnSO_4$ and Na_2SO_3) are added to the treatment circuit heads coming from the No. 1 flotation circuit. These chemicals depress most of the galena and pyrite present, which leaves the roughers as a tailing and is combined with the concentrate from the No. 2 flotation circuit. The pulp fed to this circuit contains about 20.5 to 27.4 kg Ag per metric ton (600-800 oz/ton) of concentrate. The final concentrator product has a grade of approximately 44 kg Ag per metric ton (1,294 oz/ton). This finished product from the two cleaners is high grade tetrahedrite concentrate. This concentrate is dewatered and is sent to the antimony plant for antimony removal.

No. 2 Flotation Circuit. The No. 2 flotation circuit is the same as the No. 1 flotation circuit. In this flotation circuit, Xanthate and Aerofloat 31 are added to the underflow from the No. 1 flotation circuit. This floats the remaining tetrahedrite, galena, and some of the pyrite. The concentrate from this circuit is combined with the retreatment circuit tailings and sent to the regrind circuit for upgrading.

No. 3 Flotation Circuit. The No. 3 flotation circuit is the same as the No. 1 and No. 2 flotation circuits. The purpose of this circuit is to recover the valuable minerals that are present in the No. 2 flotation circuit tailings. Copper concentrate is recovered, the final tailings produced are pumped to a sand plant, and the fines are sent to a tailings pond for final disposal. The coarse tailings are pumped to the mine, mixed with cement, and used for backfill. Analysis of the final tailings leaving the concentrator is shown in Table 114, along with background concentrations. The concentration of most elements in the tailings is considerably higher than background values, which were determined for this mining area.

Regrind Circuit. In this circuit, the low grade silver concentrate from the No. 2 flotation unit is ground to 95 percent -325 mesh in order to free the tetrahedrite that has been locked in middling particles with the quartz and the pyrite. After grinding, the material is concentrated and upgraded in two small flotation units into an end product that contains 2.7 to 3.4 kg of silver per metric ton (80-100 oz/ton).

Antimony Concentrator. The high grade silver tetrahedrite concentrate from the retreatment circuit is hauled by truck to the antimony concentrator for removal of antimony.

The tetrahedrite concentrate is leached at 100 C for about 14 hr in concentrated sodium sulfite solution in order to dissolve the antimony. This is a batch leaching operation. After leaching, the pulp is transferred to thickeners, where the solids settle to the bottom, and the clear liquor is decanted. This pregnant solution consists of antimony present as thioantimonate and is sent to an electrolysis operation. The settled solids are washed, filtered twice, and shipped to the smelter as a high grade silver-copper product.

Electrolysis. The clear pregnant solution from leaching is processed through a very complex eletrolysis operation in order to produce metallic antimony. The electrolysis process is a batch operation in which antimony is deposited on cathode plates as a brittle nonadherent layer. Every 4 days the cathodes are removed for stripping. The product produced, containing about 95 percent antimony, and 5 percent arsenic, is shipped to smelters.

In the electrolysis process, barren electrolyte is recycled to the leach plant; and anolyte solution, which has lost 50 percent of its alkalinity, is discarded to a tailings pond. This material, known as "fouled anolyte," contains 43 percent of the caustic soda entering the plant.

Mass Balance of Material. The mass balance for a typical operation is included in Figure 33. All of the wastes produced in antimony concentrating operations are pumped to tailings ponds. These wastes are the tailings from the No. 3 circuit and the fouled anolyte from the electrolysis plant. The waste from the flotation circuit is the total tailings from the flotation operations and consists of undissolved matter in the underflow (tailings) from the flotation operation.

The tailings waste consists of the minerals present in the ore. These elements are potentially hazardous if they should enter the environment. Sodium hydroxide contained in the fouled anolyte gives the tailings water a higher than normal pH and helps to prevent the metallic elements from becoming solubilized.

Identification of Potentially Hazardous Materials. The tailings from concentrating antimony ores are considered potentially hazardous. Table 111, p. 260, shows the total and potentially hazardous wastes from the mining and concentrating of antimony ores. In 1974, there were 189,000 MT dry weight (208,000 tons), and 2,303,000 MT wet weight (2,530,000 tons) of potentially hazardous wastes resulting from the concentrating antimony ores.

Tables 112, and 113, respectively, show the 1977 and 1983 projections for total and potentially hazardous wastes resulting from the mining and concentrating of antimony ores.

Waste Treatment and Disposal.

Mining Wastes and Treatment Disposal. Based on the returned questionnaires, company data, and our engineering judgment, there is no potentially hazardous waste resulting from the mining of antimony ores in the United States.

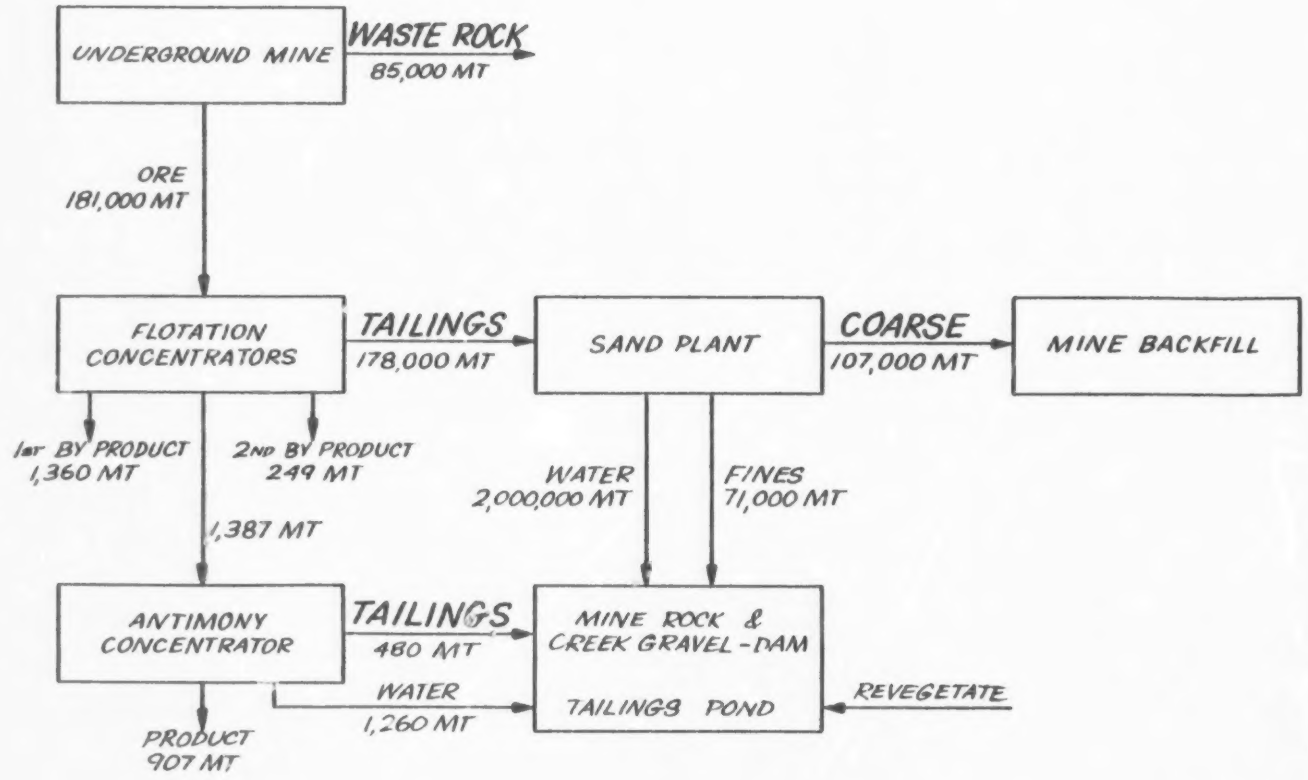
Concentrating Waste Disposal Operation. A flow diagram describing the handling of the potentially hazardous waste generated in antimony concentrating is shown in Figure 34.

Flotation. The waste from the flotation circuit consists of tailings that are made up of fine sand-like particles containing materials listed in Table 114. Note that the concentration of the mill tailings is above background levels, and therefore, considered potentially hazardous. These tailings are pumped to a sand plant and separated by cyclones into two fractions according to size. The coarse sands, which are 60 percent of the tailings, are pumped back to the mine, mixed with cement, and used to fill stopes. The fine materials are pumped to the tailings pond for final disposal.

Antimony Leach/Electrolysis. The waste from the antimony leach/electrolysis process consists of fouled anolyte containing some antimony and 43 percent of the caustic soda entering the plant. This spent anolyte solution is pumped to a tailings pond for disposal. The caustic solution is added in such quantities to raise and maintain the pH of the liquid in the tailings pond to over 8; this is beneficial in that materials considered potentially hazardous are not soluble in basic solution and settle out of the water and remain in the pond.

Waste from the flotation operation goes either to the tailings pond or back to the mine. All the waste from the leach/electrolysis process goes to the tailings pond. The tailings pond covers about 6 hectares (15 acres) and is about 3 km (2 miles) from the mill. The pond was started about 7 years ago; the dam was constructed out of mine waste and gravel from a creek bottom. There was no surface preparation for the tailings pond, except for the gravel bottom. The pond consists of dams or dikes on all four sides and is a complete pond in itself. Vegetation has been started on the dam. It consists of grasses, seedlings and elderberry bushes. The bushes, grasses, and seedlings are growing on three sides of the dam, and as the dam has just been raised, they reach about halfway up the sides of the dam.

Levels I, II, and III Technology. Level I technology for both antimony concentrating processes (flotation and leach/electrolysis) involves discharging the wastes to a tailings pond. Figure 34 shows an example of Level I technology.


UNDERGROUND MINE
WASTE ROCK
85,000 MT
ORE
181,000 MT
FLOTATION CONCENTRATORS
TAILINGS
178,000 MT
SAND PLANT
COARSE
107,000 MT
MINE BACKFILL
1st BY PRODUCT
1,360 MT
2ND BY PRODUCT
249 MT
1,387 MT
WATER
2,000,000 MT
FINES
71,000 MT
ANTIMONY CONCENTRATOR
TAILINGS
480 MT
WATER
1,260 MT
MINE ROCK & CREEK GRAVEL-DAM
TAILINGS POND
REVEGETATE
PRODUCT
907 MT

Source: Reference 8.

Figure 34. Waste from antimony operations, technology Levels I, II, and III.

TABLE 114

ANALYSIS OF TAILINGS FROM ANTIMONY MILL

Element	Concentration (ppm)	
	Mill tailings	Background
Calcium	1,887	1,500
Cadmium	1.3	-
Copper	226	21
Iron	252,550	11,800
Potassium	285	1,800
Magnesium	5,683	3,700
Manganese	20,147	490
Sodium	164	151
Lead	657	51
Antimony	437	-
Zinc	188	150

Source: Reference 7.

Level II technology consists of vegetating and stabilization of dams and dikes, surface preparation of the area before establishment of tailings ponds, and cycloning the tailings and using the coarse fraction as mine backfill. Level II technology is equivalent to Level III technology for antimony concentrating waste. Figure 34 shows an example of Levels II and III technology.

One of the two operating antimony concentrators has a Level II tailings pond and mine backfilling operation and produces 70 to 75 percent of the antimony. The other concentrating operation is new, and there is no information available to classify its disposal operations.

Based on the information supplied from industry, returned questionnaires, and our engineering judgment, there were in 1974 approximately 189,000 MT dry weight (218,000 tons) of potentially hazardous wastes resulting from the concentrating of antimony ores disposed of on land. One hundred percent of the waste is considered potentially hazardous (Table 111, p. 260).

Future Adequacy of the Technology Identified. There are no predicted changes in composition of waste as air and water pollution control facilities are installed or efficiencies improved. There may be slight changes in the volume of waste resulting from the concentrating of antimony ores.

Energy and Cost Requirements. The energy amounts and costs associated with the proposed treatment and control technologies have been estimated as a portion of the total cost necessary to implement the recommended technologies.

Waste Treatment and Disposal Costs--Antimony.

Disposal Costs - Technology Levels I, II, and III. Potentially hazardous waste disposal practices in antimony concentration utilize a tailings pond. All three levels of technology are identical for this analysis.

Similar to lead-zinc disposal practices, the antimony tailings pond is initiated with a starter dam. A sand plant is utilized to build up the dikes as the tailings accumulate. In the representative plant, the tailings pond covers 6 hectares (15 acres) and is located 3 km (2 miles) from the mill. The longer distance from pond to mill results in higher tailings pump and pipeline costs. Vegetation has also been started on the sides of the tailings dams.

Antimony waste disposal costs are presented in Table 115. The same cost assumptions as the copper cost assumptions are used. Costs per metric ton and total industry costs are presented. The total cost to the antimony mining and concentrating industry would be from $0.31 to $0.32 million per year. This would be 0.5 percent of the value added in mining and concentrating antimony ores.

TABLE 115

DISPOSAL COSTS, POTENTIALLY HAZARDOUS WASTE FROM ANTIMONY, TECHNOLOGY LEVELS I, II, AND III, TAILINGS POND (EXPRESSED IN EQUIVALENT ANNUAL 1973 DOLLARS)

	Case "A"	Case "B"
Capital		
Land (15 acres at $5,000/acre)	$ 8,775	$ 7,336
Starter dam*	4,095	4,095
Tailings pump and pipeline†	45,000	45,000
Sand plant*	6,786	6,786
Labor*	7,176	7,176
Supervision*	1,794	1,794
Maintenance*	27,061	27,061
Insurance and taxes*	11,220	11,220
Energy and power*	1,500	1,500
Vegetation‡	800	800
Total	$114,207	$112,768
Metric tons of potentially hazardous waste/year	69,398	69,398
Cost/metric ton of potentially hazardous waste disposed	$1.65	$1.63
Total cost if entire industry adopted ($$10^6$/yr)	$0.32	$0.31
Total cost as percent of value added	0.5%	0.5%

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* See major cost assumptions for copper mining.

† Estimate obtained using cost assumptions for copper mining and MRI estimate. Tailings are pumped 2 miles on representative operations.

‡ Vegetation includes dams on all four sides of ponds. Cost is MRI estimate.

Beryllium

Industry Characterization.

History of the Industry. Beryllium, discovered in 1797, was first produced in 1828 in France and Germany. Beryllium has a high melting point and is exceptionally strong and rigid, despite being nearly as light as magnesium. The hardening effect of beryllium on copper was discovered in 1926, and its industrial use for this purpose began soon afterward.

Since World War II, the unusual properties of beryllium have led to intensive research efforts to expand the supply and utilize beryllium for specialized structural applications and other purposes. Production of hand-sorted beryl in the United States is negligible, and requirements are met almost entirely from imports. Major free world producers of beryllium include Brazil, Republic of South Africa, Argentina, Australia, and Zambia.

Domestic Industry Statistics. The largest domestic source of beryllium ore is the Brush Wellman's Spor Mountain bertrandite mine near Delta, Utah. This mine contributed almost all of the U.S. mine output in 1974.

Besides the bertrandite ores in Utah, small resources of beryl are known in New England, South Dakota, and Colorado. Much larger deposits, but with beryl crystals too small for cobbing, are found in North Carolina. Deposits of bertrandite in Colorado, New Mexico, and Nevada are amenable to drilling.

Of the two beryllium companies in the United States, Brush Wellman, Inc., is fully integrated. Bertrandite ore, mined in Utah, is converted to impure beryllium hydroxide at its mill near Delta, Utah, about 76 km (47 miles) from the mine. Further refining to metal, and fabrication, occurs at Elmore, Ohio, where beryl is also processed into metal.

Use of beryllium as an alloy is increasing, but overall demand is expected to increase at an annual rate of only 2 percent through 1983.

The salient beryllium mineral statistics for the United States were withheld by the U.S. Bureau of Mines to avoid disclosing individual company confidential data.

By-Product and Coproducts. Occasionally pegmatites are mined for beryl alone, but more often beryl is recovered as a coproduct or by-product while recovering other minerals such as feldspar, mica, lithium minerals, columbite, tantalite, and cassiterite.

Waste Generation and Characterization.

Waste Generation. The land-disposed wastes generated by the beryllium industry in the United States consist of mining wastes (waste rock) and concentrator wastes (tailings). Table 109, p. 259, contains the national production data for beryllium (i.e., all statistics shown for Utah). In 1974, the total ore mined was 91,000 MT (100,310 tons), and the total beryllium products amounted to 839 MT (925 tons). The total waste rock generated was 907,000 MT (999,796 tons), and the concentrator tailings waste was 90,000 MT (99,208 tons).

Table 110 shows the ratio of waste rock and of concentrator waste to ore mined. In 1974, the ratio of waste rock to ore was 10, and the ratio of tailings to ore was 0.99.

Tables 111, 112, and 113 show the 1974 and projected 1977 and 1983 potentially hazardous waste resulting from beryllium mining and concentrating operations.

Open-Pit Mining.

Description of a Typical Process. At present there is only one operating beryllium mine in the United States. This mine employs open-pit mining methods. The mine operation is typical of open-pit mining using conventional earth-moving equipment in conjunction with drilling and blasting. The ore is loaded by shovel and hauled 76 km (47 miles) by truck to the mill. A flow diagram for the open-pit mining and concentrating of beryllium is shown in Figure 35.

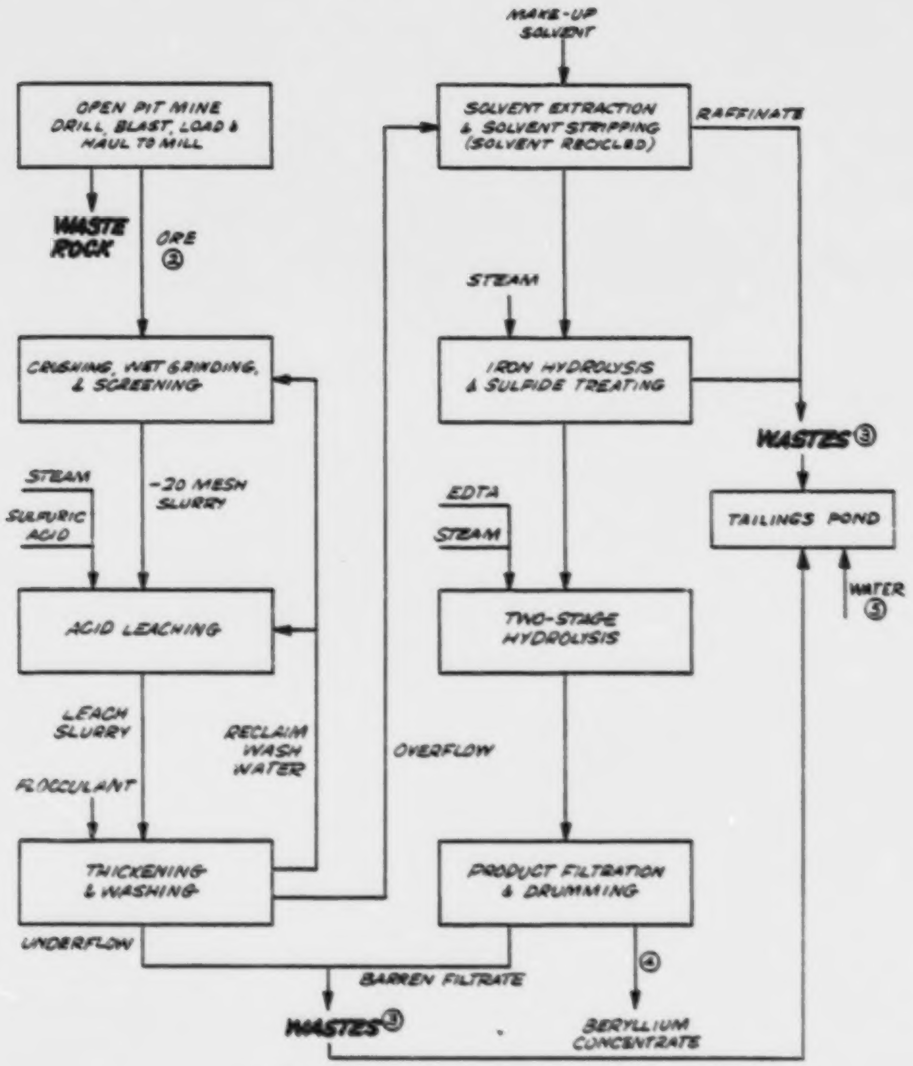
The mine waste from beryllium mining consists only of waste rock. The overburden (topsoil) and waste rock above the ore are removed by a contractor and piled along with the waste rock from mining operations in surface waste dumps.

The ore is mined from a number of small pits in a generalized area which is considered a single mine operation.

Mass Balance of Materials. The scale of operation at the beryllium mine ranges from about 70,000 to 95,000 MT (77,162-104,720 tons) of ore mined per year. The ratio of waste rock to ore is about 10:1, accounting for about 910,000 MT (1,003,103 tons) of waste generated annually. This is the only waste generated in the mining operations of beryllium ores.

Description of Individual Waste Streams. The only waste stream associated with the open-pit mining of beryllium is waste rock. This waste rock consists of rhyolite, gravel, alluvial tuff, feldspar, mica, quartz, sanidine dolomite, rhyodacite, and limestone.

Identification of Potentially Hazardous Waste Materials. The waste rock from mining operations does not contain any materials that are considered potentially hazardous.


MAKE-UP SOLVENT
OPEN PIT MINE DRILL, BLAST, LOAD & HAUL TO MILL
SOLVENT EXTRACTION & SOLVENT STRIPPING (SOLVENT RECICLED)
RAFFINATE
WASTE ROCK
ORE ②
STEAM
CRUSHING, WET GRINDING, & SCREENING
IRON HYDROLYSIS & SULFIDE TREATING
WASTES ③
STEAM
-20 MESH SLURRY
SULFURIC ACID
EDTA
STEAM
TAILINGS POND
WATER ⑤
ACID LEACHING
TWO-STAGE HYDROLYSIS
LEACH SLURRY
RECLAIM WASH WATER
OVERFLOW
FLOCCULANT
THICKENING & WASHING
PRODUCT FILTRATION & DRUMMING
UNDERFLOW
④
BARREN FILTRATE
WASTES ③
BERYLLIUM CONCENTRATE

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEMS	① WASTE ROCK*	② CRUDE ORE	③ LEACH SLUDGE SOLIDS (DRY)	④ BERYLLIUM CONCENTRATE	⑤ WATER TO TAILINGS POND
Quantity, MTPY	907,000	91,000	90,000	839	931,000
Assay, Wt. %					
Be	Present	0.5 - 0.7			
U_3O_8		Trace			
F		Present			

* Some waste rock is back filled in open pit mine.

Source: Reference 8.

Figure 35. Mining and concentrating of beryllium

Concentrator Process.

Description of Typical Process. The beryllium concentrating plant receives its ore by truck from the mining operations described earlier. The concentrating operations can be described as a tank leaching process followed by solvent extraction, hydrolysis, and precipitation.

A typical flow diagram for the concentrating of beryllium is shown in Figure 35.

Crushing and Wet Grinding. The crude ore containing about 0.5 to 0.7 percent BeO is hauled to the mill by truck where it is crushed and processed through a conventional wet grinder. The ore is wet-screened, and a -20 mesh slurry is pumped to feed tanks for the leaching operation.

Acid Leaching. The ore slurry from the grinding operations is treated with sulfuric acid, water, and heat in large leach tanks. The ore is leached continuously in the leaching circuit for about 6 hr. The pH of the leach solution is kept at about 1, and the solution is continually agitated. The solution temperature is maintained at about 93 C, and at atmospheric pressure. This continuous leach step dissolves the majority of the beryllium contained in the ore along with small amounts of other elements.

Thickening. The leachate from the leach tanks, containing some suspended solids, is removed, and flocculant is added to the solution as it enters the thickener circuit. The flocculant is added to the circuit to promote the settling of solids. The thickener circuit consists of eight thickeners employing countercurrent decantation (CCD) operations. This CCD system is necessary to separate the suspended solids from the pregnant liquor. These suspended solids would be a major problem if they were present in the solvent extraction step of the milling operation. The underflow from the thickeners is the major waste stream from the milling operation. This waste, which consists of undissolved matter (tailings) and waste liquor, is piped to a tailings pond for final disposal. The underflow from Thickeners No. 1 and No. 2 enter Thickeners No. 2 through No. 8, which are classified as wash thickeners. The wash water from Thickener No. 3 is reclaimed and used in the wet-grinding and leaching operations. The overflows from Thickeners No. 1 and No. 2 are pumped to the solvent extraction vessel. The sludge and wash water from Thickener No. 8 are pumped to the tailings pond.

Solvent Extraction. The pregnant liquor from the leaching operations is subjected to a solvent extraction (SX) process. The solvent used is a mixture of a hydrocarbon and dialkyl phosphate with the overall process being similar to uranium solvent extraction processes (alkylamines and organophosphorus compounds as solvents). The process is based on the solvent being immiscible with water and being able to form complexes with beryllium salts. These complexes are soluble in excess solvent and can be separated from the unwanted materials. In this stage, the beryllium is distributed between the aqueous and organic phases; and when conditions are controlled properly, the beryllium can be extracted quantitatively from the pregnant liquor by extraction into the organic phase. The organic phase is selective to beryllium only, with most other constituents of the leachate staying in the aqueous phase. The organic phase is separated from the aqueous phase and removed to a settling tank where the beryllium-rich organic layer rises to the surface and is separated by decantation. The barren aqueous raffinate is pumped to a tailings pond.

Solvent Stripping. The beryllium-rich organic solvent is stripped with sodium hydroxide in order to transfer the beryllium from the organic phase back to the aqueous phase. The stripped organic is processed through an acid conversion process where it is acidified with sulfuric acid, and reprocessed to the solvent extraction tank.

Iron Hydrolysis and Sulfide Treating. The aqueous solution containing sodium beryllate (Na_2BeO_2) is treated with steam in an iron hydrolysis step to produce beryllium hydroxide and sodium hydroxide (β-form). The sodium hydroxide solution is recycled to the stripping operation. The beryllium hydroxide produced is in the β-form, which is gelatinous and is difficult to filter and wash. In order to convert the $Be(OH)_2$ from the β- to the α-form, it is sulfided to beryllium sulfide and processed through a two-stage hydrolysis operation. Sludges, which are produced in both the iron hydrolysis and sulfide treating steps, are pumped to a tailings pond for disposal.

Two-Stage Hydrolysis. The beryllium sulfide from sulfide treatment is taken through a two-stage hydrolysis process in which the beryllium sulfide is oxidized by steam at about 88 to 93 C to beryllium sulfate. The sulfate is reacted with an organic chelating agent, such as a sodium salt of ethylenediaminetetraacetic acid, with addition of sodium hydroxide. The sodium beryllate which is produced is treated with steam at 160 C in the second stage to form sodium hydroxide and beryllium hydroxide (α) precipitate. This precipitate is the desired product of the concentrating operation.

Product Filtration and Drumming. The beryllium hydroxide (α) precipitate produced in the last stage of the two-stage hydrolysis is granular and is readily separated in vacuum filters. The beryllium concentrate is packed in drums and shipped by rail to Ohio.

Mass Balance of Materials. The scale of operations for the concentrating of beryllium in the United States ranges from 68,000 to 91,000 MT (74,957-100,310 tons) of ore processed each year. This amounts to about 113 to 136 MT (125-150 tons) of beryllium produced per year. The beryllium is shipped as beryllium hydroxide concentrate to Ohio. The quantity shipped each year is about 839 MT (925 tons).

Identification of Potentially Hazardous Waste Materials. The tailings from concentrating beryllium ores are considered potentially hazardous. Table 111, p.260 shows the total and potentially hazardous wastes from the mining and concentrating of beryllium ores. In 1974, there were 90,000 MT dry weight (99,000 tons), and 1,021,000 MT wet weight (1,125,000 tons) of potentially hazardous wastes resulting from concentrating beryllium ores.

Tables 112 and 113, respectively, show the 1977 and 1983 projections for total and potentially hazardous wastes resulting from the mining and concentrating of beryllium ores.

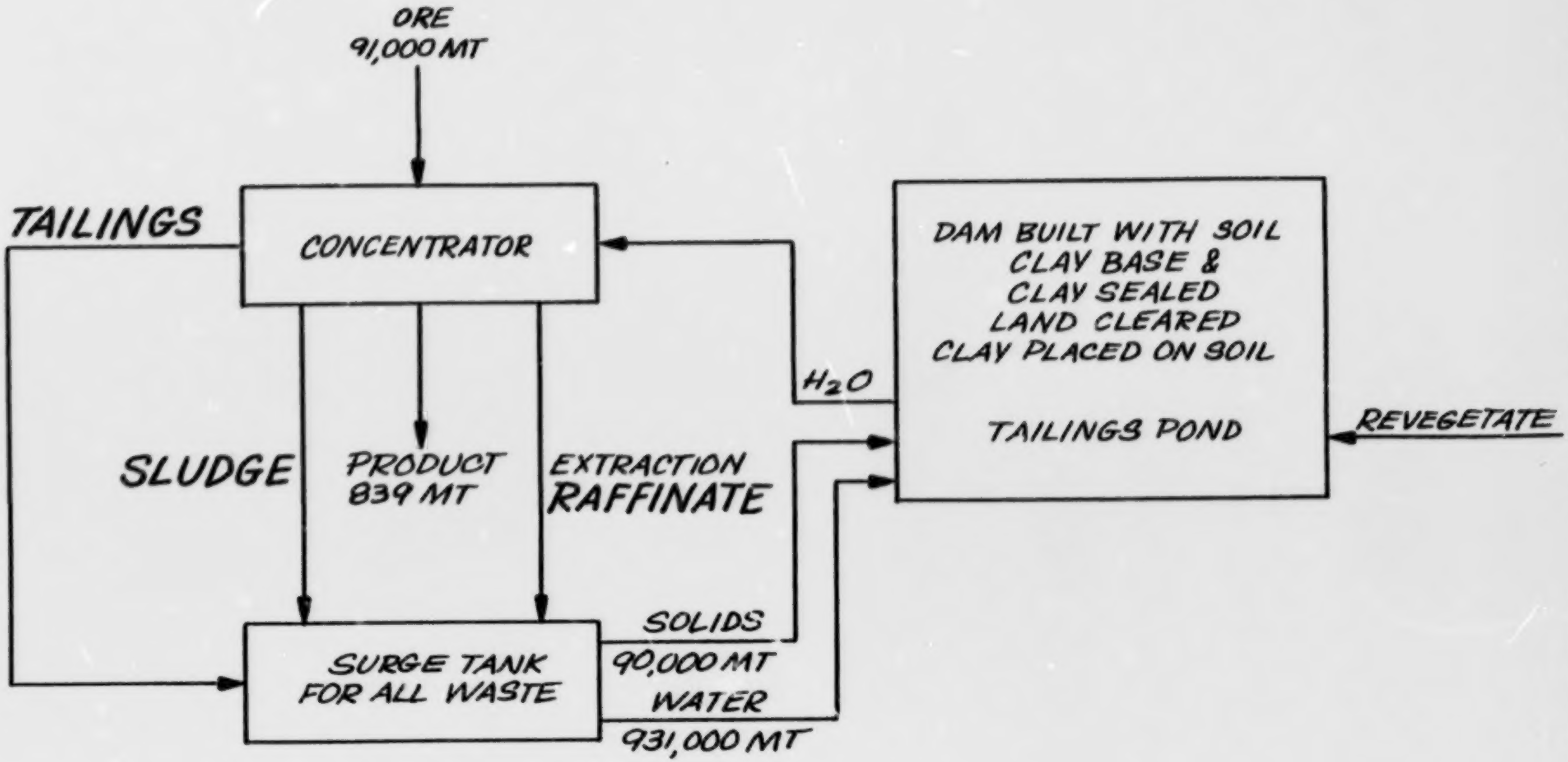
Waste Treatment and Disposal.

Mining Waste and Treatment Disposal. Based on the returned questionnaires, company data, and our engineering judgment, there were no potentially hazardous wastes resulting from the only beryllium mine in the United States.

Concentrator Waste Disposal Operations. There is only one method being used for concentrating beryllium. This method consists of an acid leaching process followed by solvent extraction, hydrolysis, and precipitation.

Acid Leaching. A flow diagram describing the handling of the potentially hazardous waste generated in beryllium concentrating is shown in Figure 36. All concentrator waste is disposed of in the tailings pond. The tailings from beryllium concentrating amount to essentially the same quantity as the ore processed per year in the mill. The waste generated in the concentrating of beryllium consists of sludges produced in the thickener circuit, the iron hydrolysis steps, and the sulfide-treating step, along with the undissolved matter (tailings) from the underflow of the thickener circuit. All wastes are combined continuously in a single tank and pumped to a common tailings pond. Of the ores processed, only 5 to 10 percent are dissolved. Thus, the leached and washed solids form the vast bulk of the solids discarded in slurry form. Based on makeup water and tonnage processed, the final solids content in slurry approximates 7 to 8 percent by weight.

The waste from concentrating averages about 125 ppm beryllium going to the tailings pond. The beryllium is measured daily, and ranges from 70 to 200 parts per million. Beryllium is the only element analyzed for in the concentrating operation. Waste from the concentrating operation would also consist of the minerals present in the beryllium ore.


ORE
91,000 MT
TAILINGS
CONCENTRATOR
SLUDGE
PRODUCT
839 MT
EXTRACTION
RAFFINATE
SURGE TANK
FOR ALL WASTE
SOLIDS
90,000 MT
WATER
931,000 MT
H_2O
DAM BUILT WITH SOIL
CLAY BASE &
CLAY SEALED
LAND CLEARED
CLAY PLACED ON SOIL
TAILINGS POND
REVEGETATE

Source: References 2 and 8.

Figure 36. Beryllium wastes--technology Levels I, II, and III.

Tailings in the tailings pond are the only source of potentially hazardous waste from beryllium concentrating. This waste could possibly contain uranium, as uranium is present in quantities very slightly higher than background in beryllium-containing ore. The uranium, as well as small amounts of beryllium that may be present in the tailings, could be hazardous if these particles should become airborne. This could occur due to wind erosion and would become a special problem during dry periods. The tailings pond in present use for disposal of tailings from the beryllium concentrating operation covers about 81 hectares (200 acres), and is divided into two sections. The pond is located about 1,500 m (4,921 ft) from the concentrator. The dike was constructed with earth and soil and has a clay base. The surface preparation for the pond consisted of clearing the vegetation and placing clay on the topsoil to seal the bottom. The pond is being stabilized by natural vegetation. The tailings pond is located in an arid region, and natural evaporation removes water from the pond, and water is recycled back to the concentrator. No water is discharged from this pond.

The use of a tailings pond is an acceptable practice to reduce potentially hazardous waste from beryllium concentrating as long as the pond is covered by water to prevent dry areas from forming. Dry areas could become a source of wind erosion which would allow tailings containing beryllium and uranium to become airborne.

Levels I, II, and III Technology. Figure 36 shows the handling of potentially hazardous beryllium concentrating wastes and shows an example of Levels I, II, and III technology. The levels of technology are identical since there is only one beryllium concentrating operation in the United States, and it meets the criteria for all three levels, namely: (1) technology currently employed by typical facilities, (2) best technology currently employed, (3) provides adequate health and environmental protection.

Level I technology for disposal of beryllium concentrating wastes is the use of tailings ponds, vegetation and stabilization of dams and dikes along with surface preparation of the area before establishment of a tailings pond. The present beryllium concentrator waste disposal system is considered Levels II, and III technology.

Based on the returned questionnaires, company data, and our engineering judgment, there were in 1974 approximately 90,000 MT dry weight (99,208 tons) of potentially hazardous waste resulting from the concentrating of beryllium ores. This represents approximately 91 percent of the concentrator waste (wet basis) going to tailings ponds.

Future Adequacy of the Technology Identified. There are no predicted changes in volume and composition of waste due to the future imposition of air and water pollution controls. The technology currently practiced at the single beryllium concentrator now operating is the total impoundment of tailings wastes. This is the best technology currently available, and it does not present any air or water pollution control problems.

Energy and Cost Requirements. The energy amounts and cost associated with the proposed treatment and control technologies have been estimated as a portion of the total cost necessary to implement the recommended technologies.

Waste Treatment and Disposal Costs--Beryllium Mines. Only one level of technology exists for the disposal of potentially hazardous wastes in the beryllium mining industry. Furthermore, the technology is only practiced in one beryllium mine in the United States. A tailings pond (covering 200 acres) is utilized. Major cost assumptions are presented in Table 116. Using the cost assumptions in Table 116, the cost of waste treatment from beryllium was calculated (see Table 117). The costs range from $0.58 to $0.54/MT of potentially hazardous waste disposed. The total cost to the industry for utilization of this technology level would be $50,000/year. This would amount to 0.5 percent of the value added.

Platinum Group Metals

Industry Characterization

History of the Industry. The first reported use of platinum was by the pre-Colombian Indians of Ecuador who made many articles of platinum and a crude platinum-gold alloy. The Spaniards gave the white infusible metal the name "platina" or little silver. Little interest was taken in the metal, chiefly because its high melting point made working it difficult. When the metal was recovered during alluvial gold mining in Colombia, it was usually discarded.

Early research was directed toward establishing methods for making the native alloy malleable. The major difficulty in the purification of platinum was due to the persistent retention of the other metals by the platinum. In 1803, Smithson Tennant recognized the presence of a new element in the insoluble residue from the aqua regia treatment of platina. Tennant named the new metal iridium. He also discovered the element osmium. Also in 1803, William H. Wollaston separated palladium and rhodium from platinum residues. And finally in 1844, the last element of the group, ruthenium, was isolated by Karl K. Klaus, a Russian chemist.

Platinum was discovered during the 1920's in Alaska, and large-scale mining of the placer began in 1934.

TABLE 116

MAJOR COST ASSUMPTIONS - BERYLLIUM MINING - TECHNOLOGY LEVELS I', II', AND III', TAILINGS POND

A. Land and construction of tailings pond = $180,000
Equivalent annual cost (no resale value) = $21,060/year

B. Other capital equipment costs = $44,000
Useful life = 10 years*
Net capital equipment cost = $60,984
Equivalent annual cost = $7,135/year

C. Annual operating costs = $19,000/year

Source: References 1 and 2.
* MRI estimate.

TABLE 117

DISPOSAL COST, POTENTIALLY HAZARDOUS WASTES, BERYLLIUM MINING, TECHNOLOGY LEVELS I, II, AND III (EXPRESSED IN EQUIVALENT ANNUAL 1973 DOLLARS)

	Case "A"*	Case "B"†
Capital		
Land‡	$21,060	$17,606
Other equipment	7,135	7,135
Operating cost	19,000	19,000
Total	$47,195	$43,741
Metric tons of potentially hazardous waste/year	81,647	81,647
Cost/metric ton of potentially hazardous waste	$0.58	$0.54
Total cost for industry ($\$10^6$/yr)	$0.05	$0.05
Total cost as percent of value added	0.5%	0.5%

* Case "A" assumes all land required is purchased in year 1 and has no resale value after 20 years.

† Case "B" assumes all land required is purchased in year 1 and resold at same price after 20 years.

‡ Land costs were obtained from individual company estimates and include land selling prices substantially less than $12,350/hectare ($5,000/acre).

Because of the useful properties and relative scarcity of these elements, they have high intrinsic value. Platinum and palladium are by far the most abundant and the most important elements of the group. In recent years, the industrial applications for platinum and palladium have greatly exceeded their use in jewelry and the decorative arts. The industrial applications for platinum and palladium are diverse, and the metals are used in the production of high-octane fuels, vitamins and drugs, electrical components, synthetic fibers, and fertilizers.

The bulk of the world supply of platinum group metals currently comes from the Soviet Union, the Republic of South Africa, and Canada.

Domestic Industry Statistics. United States mine production of platinum group metals accounted for less than 1 percent of the world output. The platinum group metals are produced as the principal product in the United States only by the Goodnews Bay Mining Company's placer operation in Alaska.

Virtually all of the remaining domestic primary metals were obtained as by-products of copper refining in Maryland, New Jersey, Texas, Utah, and Washington.

Domestic primary production of platinum in 1972 was 156 kg (543 lb). Production of platinum group metals (platinum, palladium, and rhodium) was 529 kg (1,164 lb) in 1972 and 1973 (Tables 107, p. 255, and Table 118). The U.S. demand for primary platinum in 1972 was 14,525 kg (31,960 lb). The demand for platinum group metals in the United States for existing uses is expected to increase at less than 2 percent per year through 1983. However, beginning in 1974, use as automobile emission control catalysts is expected to increase U.S. demand significantly for several years in the mid-1970's. Table 118 gives the United States platinum group metal production statistics.

By-Product and Coproduct Relationships. Gold is recovered as a coproduct or by-product with platinum metals from placer deposits. The bulk of the platinum group metals output is associated with nickel-copper minerals occurring in the ultrabasic rocks, dunite, and norite and is separated from the base metals by ore milling, smelting, and refining processes.

Waste Generation and Characterization. Platinum is being mined in Alaska by dredging methods. The ore is concentrated by gravity separation. We did not visit the Alaska facility, and no questionnaire was returned; consequently, we are unable to adequately describe the mining and concentrating process wastes.

Waste Treatment and Disposal. MRI did not assess the platinum mining and concentrating operations with regard to potentially hazardous wastes since we were unable to obtain information describing the mining and concentrating of platinum ores.

TABLE 118

U.S. PLATINUM GROUP METAL STATISTICS (KILOGRAMS)*

	1969	1970	1971	1972	1973
Mine production†	684	529	560	529	622
Refinery production					
New metal	560	622	653	466	622
Secondary metal	11,570	10,890	8,650	7,960	8,270

Source: Reference 6.

* Data reported in troy ounces and converted to kilograms.

† From crude platinum placers and by-product platinum-group metals recovered largely from domestic copper ores.

Rare Earth Metals

Industry Characterization.

History of the Industry. The rare earth elements are those 15 chemically similar elements with atomic numbers 58 to 71. The first seven members--lanthanum, cerium, praseodymium, neodymium, promethium, samarium, and europium--are commonly known as the light subgroup. The remaining eight members--gadolinum, terbium, dysprosium, holmium, erbium, thulium, ytterbium, and lutetium--comprise the heavy subgroup.

The group of elements was called the rare earths because the elements were originally considered to be scarce and because of the earth-like appearance of the oxide form. It is now realized that these elements as a whole are actually more abundant than many better known and commonly available elements.

The production of the rare earth elements in the United States, U.S.S.R., India, and Australia constitutes close to 90 percent of the world total production. U.S. demand for rare earths set a record in 1973 as consumption increased in all major uses, especially in the iron and steel industry. The growth rate was expected to decrease in 1974 because of slower economic growth and energy deficiencies, but overall consumption is expected to rise owing to increasing demand for pipeline steel, nodular iron, and petroleum catalysts. The U.S. demand for rare earths is expected to increase at an annual rate of 3 to 4 percent through 1980.

Domestic Industry Statistics. The Molybdenum Corporation of America (Molycorp) mining operation is the only rare earth producer in the United States. The mine, mill, and solvent extraction plant capacities of the Molycorp complex at Mountain Pass, California, represent about 62 percent of the total world capacity. This operation has doubled current production. Employment at this mine is estimated at nearly 100 workers. Domestic production statistics are not available because they are confidential company data.

By-Product and Coproduct Relationships. Monazite produced from beach sands and river gravel is a by-product in mining for ilmenite, rutile, and zircon. Other minerals often found associated include gold, staurolite, sillimanite, tourmaline, kyanite, andalusite, spinel, and corundum. Domestically, the rougher concentrate from the heavy sands usually contains less than 5 percent monazite; ilmenite and rutile are usually the major constituents, and zircon is next in quantity. Production from these deposits, therefore, depends primarily on the demand for titanium minerals. Low-grade monazite concentrate is recovered in the beneficiation of tungsten minerals in Colorado.

Bastnaesite ores contain no accessory minerals of great value, although barite probably could be recovered from the Mountain Pass, California, deposit.

Waste Generation and Characterization. The rare earth metal ore is being mined in California. MRI personnel were refused permission to visit the mine, and the questionnaire describing the mining and concentrating operations was not returned by the company. Information is not available in the literature to adequately describe the process and process wastes.

Waste Treatment and Disposal. MRI personnel were not permitted to visit the only existing rare earth mining and concentrating operation in the United States. Therefore, we are unable to classify the waste treatment and disposal practices resulting from mining and concentrating of rare earth ores.

Tin

Industry Characterization

History of the Industry. Pure tin was used in Egypt as early as 600 B.C. and as an alloy in bronze implements many centuries earlier. While tin touches many facets of our daily lives, it is used to improve the properties and characteristics of other materials, and its functional applications are usually hidden. It is used in coating cans, joining pipes or electrical conductors, and in bearings of all types. Tin in metal form is used in collapsible tubes, pewter ware, and pipes in the food processing industry.

Cassiterite, tin oxide, is the only commercial mineral of tin. Although cassiterite deposits are known throughout the world, those of economic importance are limited to a few areas. The principal areas are the placer deposits in Asia, and Africa, and the lode deposits in South America.

Demand for tin is expected to increase at an annual rate of about 2.2 percent through 1983.

Domestic Industry Statistics. There are no known tin deposits of economic grade or size in the United States. The United States has such small tin production capabilities as to be totally dependent upon other sources. Tin is mined by marginal operators on an interruptible basis or as a coproduct of the mining of some other materials. Small quantities of tin concentrates are produced as a by-product of molybdenum mining in Colorado, and a coproduct of placer gold mining operation in Alaska. Domestic production of tin supplied less than 0.2 percent of the total U.S. demand for primary tin in 1972. The tin mine production statistics were withheld to avoid disclosing confidential company data. Table 119 gives the U.S. production of tin, almost all of which is derived from imported ores.

By-Product and Coproduct Relationships. Tin is recovered as a by-product of molybdenum at the Climax Mine in Colorado, of lead-zinc at the Sullivan Mine in British Columbia, Canada, and as an impurity in lead refining, but this total output is an insignificant part of the U.S. production.

TABLE 119

U.S. PRODUCTION OF TIN

Year	Metric tons*
1969	350
1970	NA
1971	4,064
1972	4,368†
1973	4,572

Source: References 6 and 9.
NA = Not available.
* Data reported in tons and converted to metric tons.
† Revised.

Waste Generation and Characterization. Waste from tin mining and concentrating is not described because it is not presently being mined in the United States.

Waste Treatment and Disposal. There is essentially no mining or concentrating of tin in the United States, and consequently no potentially hazardous waste.

Titanium and Zirconium

Industry Characterization.

History of the Industry.

Titanium. Although the element titanium was discovered by William Gregor in 1790, over 100 years passed before it was put to commercial use. The first commercial application of titanium was an alloy additive to iron and steel, and the first ferroalloys were produced domestically in 1906. The metal titanium has been of commercial importance only since 1948. Titanium dioxide pigment has been produced in the United States since 1918. The use of titaniferrous materials in welding rod coating first gained acceptance in the mid-1930's.

Present technology dictates distinct end-use patterns for the two principal mineral sources of titanium, ilmenite and rutile. Ilmenite is used mostly for making titanium pigment, but rutile is used for making pigment and metal as well as a number of other important applications.

The United States is a major source of ilmenite but not of rutile. In fact, adequacy of the long-term world supply of titanium will depend to a large extent on successful development of an increased capability to use the more abundant ilmenite instead of rutile, the known reserves of which are small and of relatively high cost.

On the basis of annual titanium pigment capacity, the United States accounts for an estimated 35 percent of the world total, followed by the U.S.S.R., and other communist countries with an estimated 25 percent, followed by the United Kingdom, West Germany, Japan, and France. The United States accounts for about one-half of the world productive capacity for titanium metal, followed by U.S.S.R., Japan, and the United Kingdom.

Zirconium. Zirconium was prepared in elemental form in 1824, produced in relatively pure form in 1914, and made into high-purity grade in 1925 by the deBoer-van Arkel process. This process later was used for the first commercial zirconium production in the United States. A method of producing high-purity zirconium developed by the U.S. Bureau of Mines was adopted by private industry in 1953. Soon afterwards the zirconium industry expanded and was able to produce large quantities of high-purity and commercial-grade zirconium. Today there is considerable interest in the use of zirconium metal as structural material for nuclear reactors and chemical processing equipment. However, the consumption of the element zirconium historically has been in the form of mineral zircon, and as the oxide. Zircon is recovered only as a coproduct or by-product in connection with mining for titanium minerals and has been used extensively as facing for foundry molds, particularly in the iron and steel industry. Zirconium oxide has use in refractory applications.

Domestic Industry Statistics

Titanium. Four mines and mills were producing titanium ores in the United States during 1974. These were located in New Jersey, Virginia, and Florida. There were no data reported by the mine in Virginia, and therefore statistics for titanium in Table 109, p. 259, do not include the Virginia mine.

Demand for titanium sponge is expected to increase at an annual rate of about 8 percent through 1983. Industrial uses, as contrasted with aerospace uses, are increasing their share of titanium consumption, although at a slow rate.

Table 120 shows the production and mine shipments of titanium concentrates from domestic ores in the United States.

Zirconium. E. I. du Pont de Nemours and Company currently is the sole domestic producer of the mineral zircon, with mines and mills at Starke and Lawtey, Florida. Employment at these mines was estimated to be about 300 workers in 1974.

Demand for zirconium nonmetal and metal is expected to increase at annual rates between 3 and 5 percent through 1983. Domestic zircon resources are large, and increases in demand will probably be met by increased domestic production.

The domestic statistics concerning the production of zirconium were withheld by the U.S. Bureau of Mines to avoid disclosing confidential company data.

TABLE 120

PRODUCTION AND MINE SHIPMENTS OF TITANIUM CONCENTRATES FROM DOMESTIC ORES IN THE UNITED STATES*

Year	Production (MT)	Shipments† Quantity (MT)	Shipments† TiO_2 Content (MT)
1969	844,813	810,147	436,281
1970	787,396	835,484	442,069
1971	619,675	647,376	352,715
1972	631,153	674,410	381,822
1973	712,383	737,904	423,738

Source: Reference 9.

* As of June 30 each year.

† Data given in tons and converted to metric tons.

By-Product and Coproduct Relationships.

Titanium. Ilmenite and rutile often occur in the same deposit. Both minerals have been recovered in significant quantities from operations in Florida, and Georgia, but the two minerals were not always separated before use. Rutile and ilmenite in sand deposits may be associated with zircon, monazite, garnet, staurolite, and other heavy minerals. Most deposits are mined chiefly for rutile and ilmenite, and one or more of the associated minerals is recovered as a by-product; however, in some instances the titanium values are of secondary importance.

Zirconium. By-product mineral zircon is recovered commercially in the dredging of heavy mineral-bearing sands for ilmenite and rutile in Florida, and Georgia. Monazite, staurolite, and xenotime also were by-products in the mining operation in Georgia. Zircon, recovered as a coproduct from mining operations for titanium minerals, is imported from Australia.

Waste Generation and Characterization.

Waste Generation. The national production data for titanium-zirconium are shown in Table 109, p. 259. In 1974, the total ore mined was 22,422,000 MT (24,716,000 tons), and the total production of products amounted to 668,000 MT (736,000 tons). There was no generation of waste rock or overburden, since materials classified as waste rock in other mining operations are returned directly to the mine site. Statistics for the New Jersey operation were based on the questionnaire returned from the operating company. The State of Georgia Bureau of Mineral Statistics reported that the Georgia mines were not operating in 1974. The companies reported no waste disposal as all wastes are returned to the dredge pit, covered, and revegetated as part of the mining operation.

Mining Process. The common method used for mining of titanium and zirconium ore in this country is dredging. The ore is contained in beach sands at a concentration of about 4 percent heavy metals and is mined by dredging.

Description of Typical Process. After the surface over the ore body is cleared of trees, it is dredged. The dredge is a suction type equipped with an 18-rpm cutterhead. It has two spuds located at the stern which are used as pivots in conjunction with swinglines to advance the dredge. A 51-cm (20-in.) dredge pump powered by a 900-hp motor digs at rates up to 1,200 MT/hr (1,323 tons/hr). The presence of roots in the ored body frequently causes plug-up. To overcome this, a "root hog" was designed to chop up the roots. This is divided into two compartments by parallel bars with about 13-cm (5-in.) spacings. The dredge pump receives only the material that passes through the 13-cm (5-in.) openings. The bars start at the bottom of the box and slope upward 15° to the back. At the back, there is a combination hammermill and wood chipper consisting of a heavy rotor with hard-faced teeth that mesh between fixed dies. The rotor turns at 300 rpm, crushing the hardpan and roots until they pass through parallel bars.

The dredge pump discharges into a 61-cm (24-in.) pipeline of 12.2-m sections supported on pontoons. These connect the dredge with the floating wet mill. Two 90° ells are used to make an S-curve in the pipeline for greater dredge flexibility. Figure 37 is a flow sheet describing mining and concentrating of the ore. There are no wastes in the mining process, since all material mined is fed to the wet mill.

Mass Balance of Materials. The dredges mine at a rate of 815 to 1,090 MT/hr (898 to 1,202 tons/hr), and all material dredged is pumped directly to the wet mill where the ore is upgraded to a primary concentrate. This is shown in the table on Figure 37.

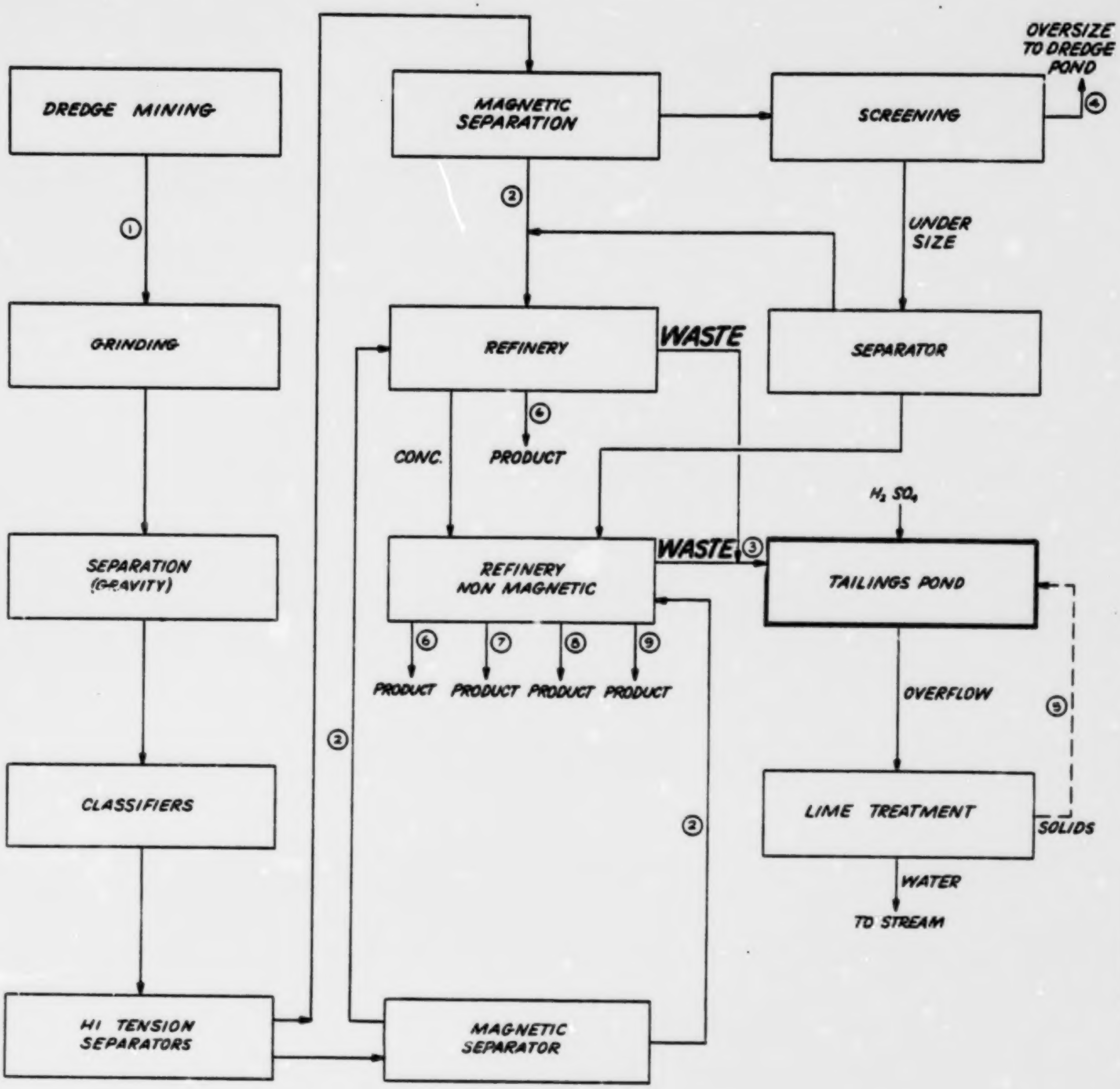
Description of Individual Waste Streams. There are no waste streams from the dredging operation.

Concentrator Process.

Wet Milling. The wet mill is constructed on three separate barges tied together as one unit. The flow diagram, Figure 37, describes the concentrating of the ore. The slurry from the dredge discharges onto two vibrating head-feed screens with 6-cm (2-in.) openings. The oversize falls by gravity into a hammermill where lumps of hardpan are disintegrated. The discharge from the hammermill flows by gravity to a screen with 2.5-cm (1-in.) openings. The screen oversize consists primarily of roots which are discharged into the pond as waste. The undersize is pumped back to the head-feed screens.

The undersize, < 0.6 cm (0.24 in.), from the head-feed screens flows by gravity to two large rougher feed sumps. The low density slurry from the dredge was originally dewatered by rake classifiers. These were later eliminated when the feed sumps were modified to serve as overflow dewatering sumps.

The wet mill treats raw ore containing about 4 percent heavy mineral at rates up to 1,090 MT/hr (1,202 tons/hr). From this raw ore it produces a concentrate averaging 85 percent heavy minerals with a recovery of 80 percent. Many of the heavy minerals lost are coarser sized silicates with lower specific gravity. Much of the TiO_2 lost is altered porous leucoxene with low apparent density.


DREDGE MINING
①
GRINDING
SEPARATION (GRAVITY)
CLASSIFIERS
HI TENSION SEPARATORS
MAGNETIC SEPARATION
②
SCREENING
OVERSIZE TO DREDGE POND
④
UNDER SIZE
REFINERY
WASTE
SEPARATOR
⑥
CONC.
PRODUCT
H₂ SO₄
REFINERY NON MAGNETIC
WASTE
③
TAILINGS POND
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PRODUCT
PRODUCT
PRODUCT
PRODUCT
OVERFLOW
⑤
②
LIME TREATMENT
SOLIDS
②
WATER
TO STREAM
MAGNETIC SEPARATOR

MATERIALS BALANCE AND COMPOSITIONS

DATA ITEM	① ORE	② FEED TO REFINERIES	③ TOTAL WASTES FROM REFINERIES	④ OVERSIZE TO DREDGE POND	⑤ FUTURE OPERATION	⑥ TOTAL TiO_2 PRODUCT	⑦ ZIRCON PRODUCT	⑧ STAUROLITE PRODUCT	⑨ ZIRCORE PRODUCT
Quantity. TPY	17.000.000	679.648	36.700	16.311.6	0	428.153	122.328	45.876	15.290
Metric TPY	15.422.145	616.566	33.294	14.798		388.414	110.974	41 618	13 871
Assay. % TiO_2	0.8 - 1.5					63	Present	Iron-Aluminum Complex	Kyanite Sillimanite Zirconium Mixture
ZrO_2	2.5 - 3.2								
Heavy Metals	1								
Lignin			100			0	0	0	
Fe						Present	Present		Present
Al						Present	Present		Present

Source: Reference 8.

Figure 37. Titanium and zirconium operations--Ti-Zr-13.

The wet mill uses three stages of concentration. The rougher stage contains 704 spirals which receive a concentrate averaging 10 to 15 percent heavy minerals; a middling containing 4 to 5 percent heavy minerals; and a tailing containing < 1 percent heavy minerals. The rougher spirals originally had five turns, but low recovery prompted the addition of two turns to each to make seven-turn roughers, with one cutter at the end of each turn. The top four turns process concentrate, and the bottom three turns process middlings. The middling is collected in four small sumps and is pumped back to the head feeder for retreatment. The tailing discharges into a large sump, from which it is pumped to the pond as backfill. Since 96 percent of the dredge feed is returned to the pond at the same rate as the dredge is mined, the dredge and the pond actually move slowly forward in the direction of mining. About every 2 weeks it is necessary to move the floating wet mill to keep up with the advance of the dredge.

The rougher concentrate is collected in four small sumps and pumped to a larger cleaner feed sump. From here it is pumped to 265 five-turn cleaner spirals. These spirals have cutters placed at the end of each turn. The upper four cutters receive a concentrate containing 40 to 50 percent heavy mineral. The bottom cutter receives a middling containing 10 to 15 percent heavy mineral. The middling flows by gravity to a small sump and is pumped back to join the rougher middlings for reprocessing in the rougher spirals. The grade of this stream is about equal to that of new feed.

The cleaner spiral concentrates flow by gravity to feed 132 three-turn finisher spirals. For control of the finisher concentrate grade, clear water is used in this stage to assist operators in making cutter adjustments. In contrast, discolored water is used in earlier stages where no cutter adjustments are made. The finisher tailings flow by gravity to the cleaner feed sump for retreatment. The concentrates are collected in two sumps and pumped by 10-cm (4-in.) pipeline to the land-based dry mill area. At times, pumping distance has been 5 km (3 miles). Booster stations are located at 460-m (1,509-ft) intervals along the pipeline. Mass balance data for this operation are contained in the flow diagram, Figure 37. Only 4 percent of the material mined is sent to the dry milling step; 96 percent of the material is put back into the dredge pond.

Dry Milling. Wet mill concentrate is a mixture of titanium minerals, heavy mineral silicates, and quartz. The function of the dry mill is to recover the titanium minerals and further separate them into an ilmenite product and a leucoxene-rutile product. The process takes advantage of the conductivity of the titanium minerals in high tension treatment, and the higher magnetic susceptibility of the ilmenite in separating the two titanium fractions.

Grain coatings on the mineral particles contribute to poor separation in the dry mill. The coatings are removed chemically by using hot sodium hydroxide.

The scrubbed ore is loaded into feed bins by a front-end loader and inclined conveyor. It is then dried by two countercurrent rotary dryers. The dryers were originally coal-fired, but were converted to fuel oil for economy. A shaking conveyor transfers the ore at 149 C from the dryers to the elevators, which distribute it to the separators. This shaker is equipped with a screen section (28 mesh) near the discharge which removes the large quartz to waste. The dry mill feed is first treated by 28 rougher high tension rolls operating in parallel. Feed is distributed by offset screw conveyors. The roughers recover about 90 percent of the titanium minerals in a concentrate containing approximately 70 percent of these minerals. A middling fraction is taken and returned to the circuit for reprocessing. The tailings are conveyed to 16 scavenger high tension rolls which recover 80 percent of the titanium mineral lost by the roughers. A middling is recirculated, and the tailings are transferred to magnets for recovery of staurolite. All products are conveyed by belt conveyors running beneath the separators. Sand temperatures up to 149 C make it necessary for the belts to be constructed of special heat-resistant rubber.

The rougher concentrates are further cleaned by 12 high tension rolls operating in parallel. The tailings from this circuit are returned to the rougher separators for retreatment. A middling is recirculated. The concentrate which contains 90 percent titanium minerals is passed over four magnets, each with two banks of five rotors operating in series. The top roll is operated at a lower flux density and scalps out the ferromagnetic tramp iron and ilmenite. The nonmagnetic fraction from each roll is fed to the next roll for separation. The magnetic fraction from all the rolls is collected and shipped as ilmenite. This product contains 98 percent titanium minerals and averages 64.5 percent TiO_2. The nonmagnetic fraction from the last roll on each magnet is collected and conveyed to a final cleaning circuit to recover the leucoxene and rutile for shipment. After the ilmenite has been removed from the cleaner concentrates, the nonmagnetic fraction contains 10 to 15 percent +45 mesh silicates. These are removed by dry screening. The undersize is then treated in four recleaner high tension separators to remove zircon and other silicates. In the original design, the tailings from this circuit were returned to the cleaner high tension circuit. It was found that this stream was low grade by comparison to rougher concentrates, which make up the bulk of the cleaner feed. These tailings often were responsible for lowering the ilmenite magnet feed grade which caused the production of low-grade material. These tailings are now returned to rougher high tension circuit for further treatment. The recleaner concentrates are finished on two high tension separators. The tailings are returned to the recleaner separators, and the high-grade concentrate shipped as a finished product. This product contains 98.5 percent titanium minerals and analyzes 80 percent TiO_2. The overall TiO_2 recovery in this dry milling operation exceeds 97 percent.

The finished products are elevated to storage bins. They are loaded from these bins into covered hopper cars for transportation to titanium dioxide pigment manufacturing plants. Cars are loaded using a timing device which opens and closes an orifice discharge device in the storage hoppers.

The tailings from the scavenger high tension circuit are fed to 10 high intensity magnets, each with three rotors operating in parallel. The magnets are equipped with a timing device which de-energizes the magnets periodically to remove tramp iron from the poles. The magnetic product consists of a mixture of staurolite with a small percentage of tourmaline, spinel, and silicates with magnetic inclusions. This product contains 45 to 50 percent Al_2O_3 and 13 to 15 percent Fe_2O_3. It is marketed for use in portland cement manufacture where it replaces the use of kaolinite as a source of Al_2O_3, and reduces the amount of slag needed to obtain the 3 percent Fe_2O_3 required in the cement. The nonmagnetic fraction from the staurolite magnets, containing 30 to 35 percent zircon, is slurried and pumped to the nearby zircon spiral plant.

Staurolite magnet tailings contain 30 to 35 percent zircon, 15 to 20 percent aluminum silicate minerals, and about 50 percent quartz. The relatively high specific gravity of zircon makes gravity separation from the lighter heavy minerals and quartz possible using spirals. The zircon spiral concentrates are dried and recleaned by a process patterned after the titanium dry mill.

The wet plant originally used three stages of spirals: 32 five-turn roughers; 32 three-turn cleaners; and 32 three-turn finishers. The tailings from the last stages were retreated in the preceding stage. The rougher spirals took a middling for recirculation. The tailings from the roughers were scavenged by 24 five-turn spirals. The scavenger concentrates were returned to the roughers, and the tailings were stockpiled for future use.

A large tonnage of zircon is used for molding sands. Many consumers require low Al_2O_3 zircon for their process, and a maximum quality specification of 1 percent Al_2O_3 was set up. This was sometimes impossible to meet with a three-stage spiral plant utilizing a scavenger circuit. As the zircon market expanded and demand increased, it became necessary to enlarge the plant. A second plant was erected using four stages of concentration. There are 32 two-turn rougher spirals. The tailings from these are remixed and fed to 32 three-turn spirals directly beneath. The concentrates from all five turns are combined to feed 32 five-turn cleaners. The tailings from these are returned to the roughers, and the concentrates are collected in a sump and fed to 32 three-turn finishers. The tailings return by gravity to the recleaner feed sump. The final wet mill concentrate, which contains 95 to 96 percent zircon, is dewatered by a screw classifier prior to drying and dry milling. After completion of the second mill, the concentrates from the original mill were retreated in the last two stages of the new mill to reduce the Al_2O_3 content to the 1 percent specification. In 1958, the original mill was redesigned and modified to be similar to the expanded mill. The two mills now operate independently of each other, and produce at the required quality.

In its natural state, the zircon has a dull brownish-purple color, caused by breakdown of crystal lattice and alteration. This color is changed to white by heating to 525 C in an oil-fired kiln. The original arrangement used an airlift to feed the kiln discharge to the dry mill. This has been replaced with a 12-m (39-ft) rotating cooler and elevator. A shaker conveyor equipped with a scalping screen is used to remove any foreign material from the cooler discharge.

Zircon wet mill concentrate contains a small percentage of titanium minerals and staurolite which must be removed. Two stages of high tension are used to separate the titanium minerals followed by magnetic separation to remove the staurolite. Zircon dry mill recovery has been increased by adding a scavenger magnet to treat the mill tailings. The scavenger concentrate is returned to the zircon magnet feed for retreatment. Some of the zircon grains lost contain blebs of magnetite which make the grains magnetic. This material is unrecoverable. Zircon specifications include a maximum of 0.15 percent Fe_2O_3 and 0.25 percent TiO_2, in addition to the 1 percent Al_2O_3. Finished zircon is stored in bins until analysis shows that it meets these specifications. It is then bagged and shipped in 45-kg (1,445-oz) bags or loaded into hopper or boxcars for bulk shipment. The shipping operation is set up to furnish the type of loading best suited to the customer's requirements. The finished product averages 98 to 99 percent zircon.

The rougher concentrates are treated on 320 five-turn rubber-lined cleaner spirals. The cleaner tailings are returned to the rougher spirals and a middling recirculated. The cleaner concentrates flow by gravity to feed 160 three-turn finisher spirals located on the deck below. These make a tailing which is returned to the cleaner spirals and a concentrate that is pumped to the dry mill area for further treatment.

Organic and clay materials are removed from the wet mill concentrates by conditioning. This consists of dewatering, scrubbing with caustic, rinsing, and final dewatering. Screw classifiers are used instead of rakes. A settling sump was incorporated to recover fines which overflow the classifiers. The scrubbed concentrate is spread by use of a high-speed slinger on the end of an inclined conveyor stacker. The slinger is directional and is used to make several small piles to allow the residual water to drain prior to drying.

A double-deck hummer screen separates the oversize from the feed to the leucoxene-rutile cleaner circuit. This serves two purposes: first, it separates the coarse (+35 mesh) staurolite; and second, it sizes the feed to the high tension separators in that circuit. The leucoxene-rutile circuit consists of four high tension rotors. One treats the -25 +80 mesh fraction from the screens. The other three treat the -80 mesh fraction. A single stage of high tension recleaning is possible because of the lower percentage leucoxene and rutile in the ore. A middling recirculates to the hummer screen. The tailings were originally designed to return to the cleaner high tension, but were later repiped to return to the rougher high tension circuit.

The wastes from the dry mill are organic matter (lignite) and kaolinite, which is pumped with the minerals from the wet mill. These wastes are harmless and are treated to settle them on the land.

Waste Treatment and Disposal. On the basis of the information collected and the mines visited we believe that there are no potentially hazardous wastes resulting from the mining and concentrating of titanium and zirconium ores.

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APPENDIX A

ACTIVE MINE AND CONCENTRATOR OPERATIONS, 1974

ACTIVE MINE OPERATIONS, 1974

1021 - Copper (56 Mines)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
American Smelting and Refining Co. Sacaton Unit Casa Grande, Arizona	1972	1	Open Pit	1	Flotation	212
American Smelting and Refining Co. Mission Operation Sahuarita, Arizona	--	1	Open Pit	1	Flotation	708
American Smelting and Refining Co. San Xavier Operation Sauharita, Arizona	1973	1	Open Pit	1	Leaching Iron Precipitation	147
American Smelting and Refining Co. Silver Bell, Arizona	1951	1	Open Pit	2	Flotation Leaching Iron Precipitation	381
American Smelting and Refining Co. Galena Unit Wallace, Idaho	1954	1	Underground	1	Flotation	200
Anaconda Company Butte, Montana	--	2	Underground Open Pit	1	Flotation	2,644
Anaconda Company Weed Heights, Nevada	1953	1	Open Pit	2	Flotation Leaching Iron Precipitation	670
Anamax Mining Company Twin Buttes Operation Sahuarita, Arizona	--	1	Open Pit	1	Flotation	1,606

ACTIVE MINE OPERATIONS, 1974

1021 - Copper (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Cities Service Company Copperhill Operation Copperhill, Tennessee	1899	5	Underground	1	Flotation	903
Cities Service Company Miami Copper Operation Miami, Arizona	1954	1	Open Pit	2	Flotation Leaching Iron Precipitation	650
Cities Service Corporation Pinto Valley Miami, Arizona	1974	1	Open Pit Solution Mining	1 1	Flotation Leaching Iron Precipitation	174
Cobre Mines, Inc. Cobre Mine Hanover, New Mexico	--	1	Open Pit	1	Leaching Iron Precipitation	22
Continental Copper, Inc. Oracle, Arizona	1974	1	Underground	1	Flotation	18
Copper Range Company White Pine Division White Pine, Michigan	1953	1	Underground	1	Flotation	2,444
Cyprus Bagdad Copper Company Bagdad Copper Pit Bagdad, Arizona	--	1	Open Pit	1	Leaching Solvent Extraction Flotation	433
Cyprus Bruce Copper and Zinc Co. Bruce Mine Division Bagdad, Arizona	1968	1	Underground	1	Flotation	562

ACTIVE MINE OPERATIONS, 1974

1021 - Copper (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Cyprus Pima Mining Company Pima Operation Tucson, Arizona	1957	1	Open Pit	1	Flotation	975
Duval Corporation Battle Mountain Operation Battle Mountain, Nevada	1967	1	Open Pit	1	Flotation	245
Duval Corporation Esperanza Operation Sahuarita, Arizona	1959	1	Open Pit	2	Flotation Leaching Iron Precipitation	542
Duval Corporation Copper Division Mineral Park Operation Kingman, Arizona	1964	1	Open Pit	1	Leaching Iron Precipitation	511
Duval Sierrita Corporation Sahuarita, Arizona	--	1	Open Pit	1	Flotation	1,339
Eagle-Picher Industries, Inc. Creta Operation Olustee, Oklahoma	1965	1	Open Pit	1	Flotation	75
Earth Resources Company Nacinjento Copper Mine Operation Cuba, New Mexico	1971	1	Open Pit	1	Flotation	116

ACTIVE MINE OPERATIONS, 1974

1021 - Copper (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
El Paso Natural Gas Company Emerald Isle Operation Kingman, Arizona	--	1	Open Pit	2	Flotation Leaching Iron Precipitation	39
Federal Resources Corporation Bonney Mine Operation Lordsburg, New Mexico	--	1	Underground	1	Flotation	113
Gold Field Corporation San Pedro Copper Mine Albuquerque, New Mexico	--	1	Underground	1	Flotation	40
Hecla Mining Company Lakeshore Mine Casa Grande, Arizona	--	1	Underground	1	Flotation	660
Inspiration Consolidated Copper Company Inspiration, Arizona	1915	3	Open Pit	3	Flotation Leaching Leach-Electrowinning	1,529
Kennecott Copper Corporation Hayden, Arizona	--		--	1	Flotation	663
Kennecott Copper Corporation Chino Mines Division Operation Hurley, New Mexico	--	1	Open Pit	1	Flotation	531

ACTIVE MINE OPERATIONS, 1974

1021 - Copper (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Kennecott Copper Corporation Ray Mines Division Ray, Arizona	1955	1	Open Pit	2	Leaching Iron Precipitation Electrowinning	811
Kennecott Copper Corporation Nevada Mines Division Ruth, Nevada	--	1	Open Pit	2	Flotation Leaching Iron Precipitation	2,284
Kennecott Copper Corporation Utah Copper Division Salt Lake City, Utah	1906	1	Open Pit	3	Flotation Leaching Iron Precipitation	4,597
Kerr American, Inc. Mammoth Mine Operation Blue Hill, Maine	--	1	Underground	1	Leaching Iron Precipitation	91
Keystone-Wallace Resources Moab Mill Operation Moab, Utah	--	1	Open Pit	1	Leaching Iron Precipitation	80
Magma Copper Company San Manuel Division San Manuel, Arizona	1956	1	Underground	1	Flotation	2,399
Magma Copper Company Superior Division Superior, Arizona	1910	1	Underground	1	Flotation	1,062
McAlester Fuel Company Zonia Operation Kirkland, Arizona	1966	1	Solution Mining	1	Leaching Iron Precipitation	24

ACTIVE MINE OPERATIONS, 1974

1021 - Copper (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Micro Copper Operation Lisbon Valley Mine Moab, Utah	--	1	Open Pit	1	Leaching Iron Precipitation	15
Phelps Dodge Corporation New Cornelia Branch Ajo, Arizona	1917	1	Open Pit	1	Flotation	633
Phelps Dodge Corporation Copper Queen Branch Bisbee, Arizona	1951 1878	2	Open Pit* Underground	1	Flotation	792
Phelps Dodge Corporation Morenci Operation, and Metcalf Mine Morenci, Arizona	1942	2	Open Pit	2	Flotation Leaching Iron Precipitation	1,079
Phelps Dodge Corporation Tyrone Branch Operation Tyrone, New Mexico	1969	1	Open Pit	2	Flotation Leaching Iron Precipitation	620
Ranchers Exploration and Development Co. Old Reliable Mine Mammoth, Arizona	1971	1	Solution Mining	1	Leaching Iron Precipitation	15
Ranchers Exploration and Development Co. Blue Bird Mine Miami, Arizona	--	1	Open Pit	1	Leaching Solvent Extraction Electrowinning	126

ACTIVE MINE OPERATIONS, 1974

1021 - Copper (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
USNR Mining and Minerals, Inc. Copper Mountain Operation Silver City, New Mexico	1970	2	Open Pit Solution Mining	1	Leaching Iron Precipitation	42
U.V. Industries, Inc. Continental Mines Operation Hanover, New Mexico	1967	2	Open Pit Underground	1	Flotation	336

* Closed in 1974.

ACTIVE MINE OPERATIONS, 1974

1031 - Lead and Zinc (47 mines)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Amax Lead Company of Missouri Buick Lead Mine Operation Boss, Missouri	1969	1	Underground	1	Flotation	289
American Smelting and Refining Co. Leadville Unit Operation Leadville, Colorado	--	1	Underground	1	Flotation	121
American Smelting and Refining Co. Ground Hog Unit Vanadium, New Mexico	--	1	Underground	1	Flotation	124
American Smelting and Refining Company Coy Mine Operations Jefferson City, Tennessee	--	1	Underground	1	Flotation	50
American Smelting and Refining Company Mascott Mine Jefferson City, Tennessee	--	1	Underground	0	Flotation	700
American Smelting and Refining Company Young Mine Operation Jefferson City, Tennessee	--	1	Underground	0	Flotation	102
American Smelting and Refining Comapny New Market Mine New Market, Tennessee	1963	1	Underground	1	Flotation	150

ACTIVE MINE OPERATIONS, 1974

1031 - Lead and Zinc (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Bunker Hill Company Bunker Hill† and Crescent Mines‡ Kellogg, Idaho	1885 1952	2	Underground	1	Flotation	1,017
Cerro Spar Corporation Salem, Kentucky	1974	1	Underground	1	Flotation	20
Clayton Silver Mine Clayton Mine Wallace, Idaho	1934	1	Underground	1	Flotation	25
Cominco American Incorporated Magmong Mine Operation Bixby, Missouri	1968	1	Underground	1	Flotation	192
Day Mines, Incorporated Day Rock Wallace, Idaho	--	1	Underground	1	Flotation	85
Eagle-Picher Industries, Inc. Shullsburg Mine Operation Shullsburg, Wisconsin	--	1	Underground	1	Flotation	87
Hecla Mining Company Star Unit Area Burke, Idaho	1940	1	Underground	1	Flotation	350
Hecla Mining Company Lucky Friday Operation Mullan, Idaho	1950	1	Underground	1	Flotation	192

† Opened in 1885.
‡ Opened in 1952.

ACTIVE MINE OPERATIONS, 1974

1031 - Lead and Zinc (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Hydro-Nuclear Corp. Linchburg Mine Magdalen, New Mexico	--	1	Underground	1	Flotation	45
Idarado Mining Company Ouray, Colorado	--	1	Underground	1	Flotation	355
Kennecott Copper Corporation Tintic Mine and Burgin Mine Eureka, Utah	1966	2	Underground	1	Flotation	286
Leadville Lead Corporation Sherman Tunnel Operation Leadville, Colorado	--	1	Underground		No Processing	75
Minerva Oil Company Minerva #1 Mine Cave In Rock, Illinois	1944	1	Underground	1	Flotation	115
New Jersey Zinc Company Eagle Mine Operation Gilman, Colorado	--	1	Underground	1	Flotation	262
New Jersey Zinc Company Sterling Hill Operation Ogdensburg, New Jersey	--	1	Underground	1	Flotation	156
New Jersey Zinc Company Friedensville Mine Operation Center Valley, Pennsylvania	--	1	Underground	1	Flotation	207

ACTIVE MINE OPERATIONS, 1974

1031 - Lead and Zinc (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
New Jersey Zinc Company Jefferson City Mine Operation Jefferson City, Tennessee	--	1	Underground	1	Flotation	126
New Jersey Zinc Company Austinville Operation Austinville, Virginia	--	2	Underground	1	Flotation	467
Ozark Lead Company Sweetwater, Missouri	1968	1	Underground	1	Flotation	264
Ozark-Mahoning Company Johnson Works Roseclare, Illinois	1937	1	Underground	1	Flotation	160
Pend Oreille Mines and Metals Company Metaline Falls, Washington	--	2	Underground	1	Flotation	92
Resurrection Mining Company Resurrection Mine Leadville, Colorado	--	1	Underground	1	Flotation	50
St. Joseph Minerals Corporation Brushy Creek Bonne Terre, Missouri	1973	1	Underground	1	Flotation	120
St. Joseph Minerals Corporation Fletcher Operation Bonne Terre, Missouri	1967	3	Underground	1	Flotation	144

ACTIVE MINE OPERATIONS, 1974

1031 - Lead and Zinc (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
St. Joseph Minerals Corporation Indian Creek #23 Bonne Terre, Missouri	1954	1	Underground	1	Flotation	114
St. Joseph Minerals Corporation Viburnum Operations Bonne Terre, Missouri	1960	2	Underground	1	Flotation	266
St. Joseph Minerals Corporation Balmat-Edwards Mine Operations Edwards, New York	--	2	Underground	1	Flotation	501
E. G. Sommerlath Enterprises Marion, Kentucky	--	1	Open Pit	1	Extraction Heavy Media	20
Standard Metals Corporation Sunnyside Operation Silverton, Colorado	--	1	Underground	1	Flotation	126
U.S. Steel Corporation Zinc Mine Jefferson City, Tennessee	--	1	Underground	1	Flotation	200
U.V. Industries Princess Mine and Hanover Mine Hanover, New Mexico	--	2	Underground	1	Flotation	400

ACTIVE MINE OPERATIONS, 1974

1092 - Mercury (2 mines)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Mine Management Company Knoxville Mine Napa County, California	1970	1	Open Pit	1	Roasting Condensing	5
One Shot Mining Company Manhattan Mine Napa County, California	1968	1	Open Pit	1	Roasting Condensing	15

ACTIVE MINE OPERATIONS, 1974

1094 - Vanadium and Uranium (158 mines and 17 concentrators)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Anaconda Company Jackpile Paguate Unit Corporation Grants, New Mexico	--	2	Underground Open Pit	1	Alkaline Leaching Ion Exchange	597
Atlas Corporation Big Indian Mine Moab, Utah	--	1	Underground	1	Dual Process Acid Leaching Plus Alkaline Leaching	89
Continental Oil Company - Pioneer Nuclear Conquista Project Falls City, Texas	1973	4	Open Pit	1	Acid Leaching	357
Cotter Corporation Cotter Mill Operation Canon City, Colorado	--	No Mine	--	1	Dual Process Acid Leaching Plus Alkaline Leaching	77
Cotter Corporation Schwartzwalder Operation Colorado	1960	1	Underground	0	No Processing	83
Dawn Mining Company Well Pinit Mine Ford, Washington	--	1	Open Pit	1	Acid Leaching Ion Exchange	250
Exxon Company USA Highland Uranium Operation Casper, Wyoming	1972	1	Open Pit	1	Acid Leaching Solvent Extraction	83

ACTIVE MINE OPERATIONS, 1974

1094 - Vanadium and Uranium (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Exxon Corporation Exxon Company, USA Division Fall City, Texas	1970	1	Open Pit	0	No Processing	55
Federal American Partners Riverton, Wyoming	1969	2	Open Pit Underground	1	Acid Leaching Ion Exchange Solvent Extraction	75
Four Corners Exploration Company Dog Mine Grants, New Mexico	--	2	Underground	0	No Processing	10
Homestake Mining Company F33 Mine Grants, New Mexico	--	1	Underground	0	No Processing	15
Kerr-McGee Corporation Grants, New Mexico	--	10	Underground	1	Acid Leaching Solvent Extraction	1,200
Kerr-McGee Corporation* Shirley Basin Operation Casper, Wyoming	--	1	Open Pit	0	No Processing	124
Mountain West Mine, Inc. Betty Mine Blanding, Utah	--	1	Open Pit	0	No Processing	19
Petrotomics Company & K.G.S.--J.V. Joint Venture Mine Casper, Wyoming	1960	1	Open Pit	1	Acid Leaching Solvent Extraction	85

ACTIVE MINE OPERATIONS, 1974

1094 - Vanadium and Uranium (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Ranchers Exploration and Development Corp. Grants, New Mexico	--	1	Open Pit	0	--	15
Rio Algom Corporation Lisbon Mine Operation Moab, Utah	1972	2	Underground	1	Alkaline Leaching	172
Susquehanna-Western, Inc. Falls City, Texas	1970	1	Open Pit	0	No Processing	35
Union Carbide Corporation Wilson Springs Operation Hot Springs National Park, Arkansas	1969	3	Open Pit	1	Salt Leaching Solvent Extraction for Vanadium Concentrate	177
Union Carbide Corporation Uravan Operation Uravan, Colorado	--	25	Underground	1	Acid Leaching Ion Exchange Vanadium Recovery	293
Union Carbide Corporation Riverton, Wyoming	1972	2	Open Pit	1	--	118
United Nuclear Corporation Grants, New Mexico	--	3	Underground	0	--	405
United Nuclear-Homestake Partners Grants, New Mexico	1958	4	Underground	1	Alkaline Leaching Solvent Extraction	450
Utah International, Inc. Shirley Basin Operation Casper, Wyoming	--	1	Open Pit	1	Acid Leaching Ion Exchange	474

ACTIVE MINE OPERATIONS, 1974

1094 - Vanadium and Uranium (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Utah International, Inc. Lucky McMine Operation Riverton, Wyoming	1957	2	Open Pit Underground	1	Acid Leaching Ion Exchange	351
Western Nuclear Rox Operation Casper, Wyoming	--	1	Open Pit	0	--	150
Western Nuclear, Inc. Golden Goose Operation Jeffrey City, Wyoming	--	1	Open Pit	1	Acid Leaching Ion Exchange Solvent Extraction	151
Other unidentified uranium mines (including 37 in Colorado)	--	83	Open Pit Underground	0	--	267

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* Operations at this site were suspended indefinitely in November, 1974.

ACTIVE MINE OPERATIONS, 1974

1099 - Miscellaneous Mining (9 mines)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
American Smelting and Refining Co. Manchester Mine Lake Hurst, New Jersey	--	1	Dredge	1	Gravity	80
Brush Wellman, Inc. Delta Mill Operation Delta, Utah	1968	--	--	1	Leaching Solvent Extraction Precipitation	52
Brush Wellman, Inc. Spar Mountain Operation Delta, Utah	1968	1	Open Pit	0	--	11
E. I. duPont Highland Mine Lawtey, Florida	1952	1	Dredge	2	Electrostatic Magnetic and Gravity Separation	150
E. I. duPont Trailridge Mine Starke, Florida	1952	1	Dredge	2	Electrostatic Magnetic and Gravity Separation	150
Felspar Corporation Montpelier Mine Montpelier, Virginia	--	1	Open Pit	1	Electrostatic Magnetic and Gravity Separation	41
Goodnews Bay Mining, Company Platinum, Alaska	--	1	Dredge	1	Gravity	37
Molybdenum Corporation of America Mountain Pass Operation Nipton, California	--	1	Open Pit	1	Flotation	93

ACTIVE MINE OPERATIONS, 1974

1099 - Miscellaneous Mining (continued)

Name of Operation	Year Opened	Number of Mines	Mine Type and Process	Number of Concentrators	Concentration Process	Estimated Employment
Sunshine Mining Company Big Creek Mine Kellogg, Idaho	1960	1	Underground	2	Flotation Hot Leaching	700
U.S. Antimony Corporation Babbitt Operation Thompson Fall, Montana	1971	1	Underground	1	Flotation Heavy Media Separation	14

APPENDIX B

MINES AND CONCENTRATORS VISITED

QUESTIONNAIRES RECEIVED

SAMPLE QUESTIONNAIRE

TABLE 8-1

MINES VISITED

Company Name	Mine Name	Location		SIC No.	Mine		Concentrator	
		City	State		No.	Type	No.	Type
1. Mine Management Company	Knoxville	Napa County	California	1092	1	Open Pit	1	Roast and Condense
2. One Shot Mining Company	Manhattan	Napa County	California	1092	1	Open Pit	1	Roast and Condense
3. Amax Lead Company of Missouri	Buick	Buick	Missouri	1031	1	Underground	1	Flotation
4. St. Joe Minerals Company	Brushy Creek	Bonne Terre	Missouri	1031,1021	1	Underground	1	Flotation
St. Joe Minerals Company	Fletcher	Bonne Terre	Missouri	1031,1021	2	Underground	1	Flotation
St. Joe Minerals Company	Indian Creek	Bonne Terre	Missouri	1031,1021	1	Underground	1	Flotation
St. Joe Minerals Company	Viburnum	Bonne Terre	Missouri	1031,1021	3	Underground	1	Flotation
5. Idarado Mining Company	Idarado	Ouray	Colorado	1031,1021	1	Underground	1	Flotation
6. United States Steel Company	Davis-Bible	Jefferson City	Tennessee	1031	1	Underground	1	Flotation
7. Kennecott Copper Corporation	Bingham	Salt Lake City	Utah	1021,1099	1	Open Pit	3	2 Flotation, 1 Leach Precipitation
8. Anamax Mining Company	Twin Buttes	Sahuarita	Arizona	1021	1	Open Pit	2	1 Flotation, 1 Leach Precipitation
9. Cyprus Pima Mining Company	Pima	Sahuarita	Arizona	1021	1	Open Pit	1	Flotation
10. McAlester Fuel Company	Zonia	Kirkland	Arizona	1021	1	*In Situ*	1	Leach - Precipitation
11. Phelps Dodge Corporation	Copper Queen-				1	Underground	1	Flotation
	Lavender Pit	Bisbee	Arizona	1021	1	Open Pit	1	Leach - Precipitation
12. Magma Copper Company	Magma	Superior	Arizona	1021	1	Underground	1	Flotation
13. Kennecott Copper Corporation	Ray	Hayden	Arizona	1021	1	Open Pit	1	Leach - Precipitation
							1	Leach - Electrowinning
							1	Flotation
14. Ranchers Exploration and Development Corp.	Bluebird	Miami	Arizona	1021	1	Open Pit	1	Leach - Solvent Exchange - Electrowinning
15. Kennecott Copper Corporation	Burgin	Eureka	Utah	1031	2	Underground	1	Flotation
	Trixie							
16. Brush Wellman	Topaz	Delta	Utah	10[illegible]	1	Open Pit	1	Acid Leach - Solvent Extraction - Hydrolysis
17. E. I. duPont	Trail Ridge	Starke	Florida	1099	1	Dredge	2	Electrostatic - Magnetic Separation - Wet
	Highland	Lawtey	Florida	1099	1	Dredge	2	Dry - Electrostatic - Magnetic Separation
18. Bunker Hill Company	Bunker Hill	Kellogg	Idaho	1031	1	Underground	1	Flotation
	Crescent			1021	1	Underground		
19. Sunshine Mining Company	Sunshine	Kellogg	Idaho	1021,1031 1099	1	Underground	2	Flotation (Antimony Plant)
20. Union Carbide Corporation	North Wilson East Wilson Spaulding	Hot Springs	Arkansas	1094	3	Open Pit	1	Saltroast - Water Leach Precipitation
21. Cities Service Company	Calloway Boyd Cherokee Tennessee Eureka	Copper Hill	Tennessee	1021	5	Underground	1	Flotation
22. The Anaconda Company	Jack Pile-Paguate	Grants	New Mexico	1094	1	Open Pit	1	Alkaline Leach - Precipitation
23. United Nuclear - Home Stake Partners	Ambrosialake Operations	Grants	New Mexico	1094	4	Underground	1	Roast - Leach - Filtration - Precipitation
24. Cotter Corporation	Schwartzwalder	Canon City	Colorado	1094	1	Underground	1	Leach - Autoclave - Filtration - Precipitation
25. Union Carbide Corporation	--	Uravan	Colorado	1094	3	Underground	1	Leach - Ion Exchange - Precipitation
26. Exxon Company	Highlands	Casper	Wyoming	1094	1	Open Pit	1	Leach - Solvent Extraction
27. Utah International, Inc.	--	Shirley Basin	Wyoming	1094	1	Open Pit	1	Leach - Ion Exchange

TABLE B-2

QUESTIONNAIRES RECEIVED

				Mine			Concentrator	
	Company Name	Location	SIC No.	Name	No.	Type	No.	Type
1.	Idarado Mining Company	Telluride, Colorado	1031,1021	Idarado	1	Underground	1	Flotation
2.	Union Carbide	Hot Springs, Arkansas	1094	North Wilson, East Wilson, Spaulding	3	Open Pit	1	Saltroast - Solvent Extraction
3.	Conquista Project	Falls City, Texas	1094	Conquista Project	4	Open Pit	1	Hydrometallurgy
4.	Phelps Dodge	Morenci, Arizona	1021	Morenci-Metcalf	2	Open Pit	2	Flotation - Leach Precipitation
5.	Phelps Dodge	Bisbee, Arizona	1021	Copper Queen-Lavender Pit	2	1-Open Pit 1-Underground	1	Flotation
6.	Phelps Dodge	Ajo, Arizona	1021	New Cornelia	1	Open Pit	1	Flotation
7.	Phelps Dodge	Tyrone, New Mexico	1021	Tyrone	1	Open Pit	2	Flotation - Leach Precipitation
8.	Bunker Hill Company	Kellogg, Idaho	1031	Bunker Hill - Crescent	2	Underground	1	Flotation
9.	Cotter Corporation	Canon City, Colorado	1094	Schwartzwalder	1	Underground	1	Leach - Auto Clave
10.	Continental Copper	Oracle, Arizona	1021	Oracle Ridge Project	1	Underground	1	Flotation
11.	Cominco America, Inc.	Bixby, Missouri	1031	Magmont	1	Underground	1	Flotation
12.	Utah International, Inc.	Riverton, Wyoming	1094	Lucky Mc	1	Open Pit	1	Acid Leach Ion Exchange
13.	Cities Service Company	Copperhill, Tennessee	1021,1031	Calloway-Boyd, Cherokee Tennessee-Eureka	5	Underground	1	Flotation
14.	Amax Lead Company	Boss, Missouri	1031	Buick	1	Underground	1	Flotation
15.	Petrotomics Company	Casper, Wyoming	1094	Shirley Basin	1	Open Pit	1	Leach - Solvent Extraction
16.	Day Mines, Inc.	Wallace, Idaho	1031	Dayrock	1	Underground	1	Flotation
17.	Exxon Company	Casper, Wyoming	1094	Highland Operations	1	Open Pit	1	Acid Leach Solvent Extraction
18.	Kennecott Copper	Bingham Canyon, Utah	1021	Bingham	1	Open Pit	3	Flotation(2) - Leach Precipitation
19.	Cyprus Pima Mining Company	Tucson, Arizona	1021	Pima	1	Open Pit	1	Flotation
20.	The Anaconda Company	Grants, New Mexico	1094	Jackpile-Paguate	2	Open Pit Underground	1	Leach - Resin in Pulp
21.	American Smelting and Refining Company	Immel, Tennessee	1031	Young-Coy-Immel	3	Underground	1	Flotation
22.	American Smelting and Refining Company	Jefferson County, Tennessee	1031	New Market	1	Underground	1	Flotation
23.	American Smelting and Refining Company	Wallace, Idaho	1021	Galena	1	Underground	1	Flotation
24.	American Smelting and Refining Company	Sahuarita, Arizona	1021	San Xavier	1	Open Pit	1	Acid Leach Precipitation
25.	American Smelting and Refining Company	Sahuarita, Arizona	1021	Mission Unit	1	Open Pit	1	Flotation
26.	American Smelting and Refining Company	Lake Hurst, New Jersey	1099	Manchester	1	Dredge	1	Gravity
27.	American Smelting and Refining Company	Deming, New Mexico	1031	Ground Hog-Deming	2	Underground	1	Flotation
28.	American Smelting and Refining Company	Casa Grande, Arizona	1021	Sacaton Unit	1	Open Pit	1	Flotation
29.	American Smelting and Refining Company	Silver Bell, Arizona	1021	Silver Bell	1	Open Pit	2	Flotation - Leach Precipitation
30.	Copper Range Company	White Pine, Michigan	1021	White Pine	1	Underground	1	Flotation
31.	The New Jersey Zinc Company	Friedensville, Pennsylvania	1031	Friedensville	1	Underground	1	Flotation
32.	The New Jersey Zinc Company	Ivanhoe, Virginia	1031	Austinville-Ivanhoe	2	Underground	1	Flotation
33.	The New Jersey Zinc Company	Jefferson City, Tennessee	1031	Jefferson City	1	Underground	1	Flotation
34.	The New Jersey Zinc Company	Gilman, Colorado	1031	Eagle	1	Underground	1	Flotation
35.	Gulf Resources and Chemical Corporation	Metaline Falls, Washington	1031	Pend Oreille	1	Underground	1	Flotation
36.	Kennecott Copper	Ray, Arizona	1021	Ray	1	Open Pit	3	Flotation - Leach Precipitation Leach Electrowinning
37.	Magma Copper Company	San Manuel, Arizona	1021	San Manuel	1	Underground	1	Flotation
38.	Magma Copper Company	Superior, Arizona	1021	Magma	1	Underground	1	Flotation
39.	Union Carbide Corporation	Gas Hills, Wyoming	1094	Gas Hills District	2	Open Pit	1	Leach Ion Exchange Precipitation
40.	Union Carbide Corporation	Uravan, Colorado	1094	Uravan	3	Underground	1	Leach Ion Exchange Precipitation
41.	Hecla Mining Company	Burke, Idaho	1031	Star Unit Area	1	Underground	1	Flotation
42.	Hecla Mining Company	Mullan, Idaho	1031	Lucky Friday	1	Underground	1	Flotation
43.	Hecla Mining Company	Casa Grande, Arizona	1021	Lakeshore Project	1	Underground	1	Flotation
44.	Cyprus Mines Corporation	Bagdad, Arizona	1021,1031	Cyprus	1	Underground	1	Flotation
45.	Brush Wellman	Delta, Utah	1099	Topaz	1	Open Pit	1	Hot Leach - Solvent Extraction Precipitation

Revised 10/15/74

(SAMPLE QUESTIONNAIRE)

WASTE DISPOSAL BACKGROUND INFORMATION

(for MRI Project No. 3952-D)

1. Corporation Name: ______________________________

____

Mailing Address: ______________________________

____

____

Mine Names and Locations: 1. ______________________________

2. ______________________________

3. ______________________________

4. ______________________________

2. Person to contact regarding information supplied herein:

Mr./Ms. ______________________________

Address ______________________________

____

Phone ______________

3. Final Concentrator Product(s):

(primary)____________________TPY* ____________________; Grade,%________

(byproduct)__________________TPY ____________________; Grade,%________

(byproduct)__________________TPY ____________________; Grade,%________

4. Mine Operations:

(a) Type: Open Pit_________; Underground_________; Age of Mine______

(b) Water (from mine operation only) discharged to tailing ponds--describe source(s)____

____

Gallons per day:______________________; pH______________________

(c) Solids handled:

Ore- ____TPY

Solid Waste (waste rock and/or overburden)______________________TPY
Percent waste rock__________________; Percent overburden__________

5. Leaching Operations (if applicable):

(a) Type: Dump___________; Heap___________; Vat_________; Waste_____;

Other (specify)___________________________

(b) Solvent: Acid__________; Alkali______________; Cyanide__________;

Other (specify)___________________________

6. Concentrating Operations:

(a) Grind: Percent above 100 mesh________; Percent below 100 mesh________

(b) Process used (check); Froth Flotation ______; Gravity_______;

Other (specify)___________________________

(c) Volume of water in circuit:______________________gallons per minute

(d) Water discharged to tailings ponds--describe source(s)_______________

____

Gallons per day:__________________________

---

*TPY=tons per year

7. Solid Waste Disposal:

Source #1 ______________________________
TPY (dry short tons)______________________________
Tailing impoundment____________; Area (Acres)__________
Mine backfill______________; Other (specify)__________

Source #2 ______________________________
TPY (dry short tons)______________________________
Tailing impoundment____________; Area (Acres)__________
Mine backfill______________; Other (specify)__________

Source #3 ______________________________
TPY (dry short tons)______________________________
Tailing impoundment____________; Area (Acres)__________
Mine backfill______________; Other (specify)__________

Source #4 ______________________________
TPY (dry short tons)______________________________
Tailing impoundment____________; Area (Acres)__________
Mine backfill______________; Other (specify)__________

(Indicate additional sources on back of sheet)

8. What waste disposal techniques do you use for all of your wastes? Please check techniques used, and percentage of the total waste treated by the method.

	Check Method	Percent of Total Waste
Burial (deep)	____	__________
Burial (surface)	____	__________
Detoxification (chemical and biological)	____	__________
Recovery and Reuse	____	__________
Deep Well Injection	____	__________
Mine Disposal	____	__________
Open Burning	____	__________
Incineration	____	__________
Open Dumping	____	__________
Tailing Pond	____	__________
Other (specify)	____	__________

9. Do you handle your own waste disposal? ________Yes ________No

10. Percent of solid waste sold ______________; used for_________________

11. Liquid discharged to tailing ponds or other land disposal (characteristics in milligrams per liter).

	Suspended Solids	Total Dissolved Solids	TDS, Major Constitutent
Discharge #1			
Discharge #2			
Discharge #3			
Discharge #4			

(list additional discharges on back of sheet)

12. Solid Waste Characteristics: (see next page)

13. Approximate cost for treatment and land disposal of wastes $_____/ tons of waste

11. Solid Waste Characteristics: (attach additional sheets for more than four sources)

Indicate, by a check mark, which of the following are analyzed for: in the land-disposed solid wastes; show the sensitivity level (in parts per million) of the analytical technique used, and the test frequency (D = Daily, W = Weekly, M = Monthly, Y = Yearly).

	Asbestos	Arsenic	Beryllium	Cadmium	Chromium	Copper	Cyanides	Mercury	Pesticides	Selenium	Zinc	Lead	Radioactivity (Pci)**
Source #1 (Analyzed for, check)													
Concentration, ppm*													
Average													
Range													
Test Sensitivity, ppm													
Test Frequency													
Source #2 (Analyzed for, check)													
Concentration, ppm													
Average													
Range													
Test Sensitivity, ppm													
Test Frequency													
Source #3 (Analyzed for, check)													
Concentration, ppm													
Average													
Range													
Test Sensitivity, ppm													
Test Frequency													
Source #4 (Analyzed for, check)													
Concentration, ppm													
Average													
Range													
Test Sensitivity, ppm													
Test Frequency													

* Parts per million
** pico curie per liter

APPENDIX C

GEOLOGY AND MINERALOGY

GEOLOGY AND MINERALOGY

Copper Ores

Copper ores occur in rocks of nearly all kinds and ages and in ore deposits of many types. All copper ores are secondary deposits and can be classified geologically into a multitude of types, but in this discussion they will be grouped into two classes--disseminated "porphyry," and all others. The disseminated deposits, usually containing less than 1 percent copper, are large masses of more or less altered and decomposed igneous rock, in which copper is uniformly, though sparsely, distributed in the form of small particles and veinlets; the rock usually is of a granitic type and porphyritic texture, hence the term "porphyry copper." Other types of deposits include bedded deposits, vein deposits, massive replacement deposits, etc. The practical importance of the porphyry type is that its size and physical character encourage application of large-scale, low-cost mining, and metallurgical methods.

The principal copper minerals as they occur in each state where copper is mined are listed in Table C-1. Of the sulfide ores listed, bornite, chalcopyrite, and enargite are considered the "primary" minerals which are redeposited by igneous processes deep in the earth's crust. Minerals such as covellite and chalcocite were largely formed as "secondary" deposits of copper leached from the sulfides close to the surface and precipitated near the water level. The oxide minerals such as chrysocolla, malachite, and azurite were formed through oxidation of the surface sulfides. Native copper is chiefly located in the oxidized zones of an ore deposit; however, this is not the case in the major native copper deposits in Michigan, which are generally considered to be "primary" in source.1/ The copper at White Pine, Michigan, from which a little more than 5 percent of the United States primary copper currently is produced, is a large stratiform ore body 1.2 to 7.6 meters thick and several kilometers across. The present ore column, containing about 1.2 percent as chalcocite and native copper, is confined to the basal beds of the Nonsuch Shale. This upper Keweenawan (upper Precambrian) formation is a distinctive gray siltstone unit about 183 meters thick, overlying and overlain by thick red-bed sequences. All these rocks contain abundant volcanic debris, but the latest known igneous activity within 80 kilometers antedated the Nonsuch Shale. The rocks are unmetamorphosed and only moderately deformed, mainly by tilting and faulting.2/

Table C-1

SUMMARY OF MINERALOGICAL SYSTEMS
COPPER ORE AT MINES (1021)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
Arizona and adjacent New Mexico	Ankerite	$2CaCO_3 \cdot MgCO_3 \cdot FeCO_3$	2.9-3.2	s. a.	-	3.5-4
	Barite	$BaSO_4$	4.50	s. a.	i.	3-3.5
	Bornite (bn)	Cu_5FeS_4	5.06-5.08	s. HNO_3	sl. s.	3
	Chalcocite (cc)	Cu_2S	5.5-5.8	s. HNO_3	-	2.5-3
	Chalcopyrite (cp)	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Tennantite	$5Cu_2Si2ZnS \cdot 2As_2\text{-}S_3$	4.37-4.49	d HNO_3	-	3-4
	Enargite (en)	Cu_3AsS_4	4.4-4.5	s. HNO_3	-	3
	Galena (gn)	PbS	7.57-7.59	dec. HNO_3	-	2.5-2.75
	Tetrahedrite	$3Cu_2S \cdot SloS_2 \cdot Fe \cdot Zn\text{-}Ag$	4.4-5.1	dec. HNO_3	-	3-4
	Hematite (hm)	Fe_2O_3	5.26	s. a.	i.	5-6
	Magnetite (mg)	Fe_3O_4	5.16	s. HCl	-	5.5-6.5
	Pyrite (py)	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Quartz (qz)	SiO_2	2.65	s. only HF	i.	7
	Sphalerite (sl)	ZnS	3.9-4.1	s. HCl	i.	3.5-4
	Azurite	$2CuCO_3 \cdot Cu(OH)_2$	3.4-3.8	s. a.	i.	3.5-4
	Chrysocolla	$CuSiO_3 \cdot H_2O$	2-2.24	s. a.	-	2
	Covellite	CuO	4.6	s. HNO	-	1.5-2
	Serpentine	$3MgO_2(SiO_2) \cdot 2H_2O$	2.5-2.65	dec. a.	i.	2.5-3.5
	Cuprite	Cu_2O	5.8-6.2	s. HCl	i.	3.5-4
Tennessee	Halloysite	$Al_2O_3 \cdot SiO_2 \cdot H_2O$	2.0-2.2	i.	i.	1-2
	Hemimorphite	$Zn_4Si_2O_7(OH)_2 \cdot H_2O$	3.4-3.5	gel. a.	i.	5
	Limonite	Hydrated ion oxides	3.6-4	s. HCl	i.	5-5.5
	Malachite	$CuCO_3 \cdot Cu(OH)_2$	4.00	s. a.	dec.	3.5-4
	Manganite	$Mn_2O_3 \cdot H_2O$	5.16	s. HCl	-	5.5-6.5
	Olivenite	$4CuO \cdot As_2O_5 \cdot H_2O$	4.1-4.4	s. HNO_3	i.	3
	Psilomelane	$MnO_2 \cdot Ba \cdot K_2 \cdot Na_2O \cdot H_2O$	3.3-4.7	s. HCl	i.	5.7
	Tremolite	$Ca_2Mg_5Si_8O_{22}\text{-}(OH_1F)_2$	3.0	-	i.	5-6
	Anhydrite	$CaSO_4$	2.8-3.0	s. HCl	i.	3-3.5
	Pyrrhotite	$Fe_{11}S_{12}$	4.58-4.65	dec. a.	sl. dec.	3.5-4.5
	Cubanite	$CuFe_2S_3$	4.03-4.18	s. a.	sl. s.	3.5
	Molybdenite	MoS_2	4.62-4.73	s. a.	i.	1-1.5
	Gypsum	$CaSO_4 \cdot 2H_2O$	2.32	s. HCl	i.	1.5-2
	Pyrrhotite	$Fe_{11}S_{12}$	4.58-4.65	dec. a.	sl. dec.	3.5-4.5
	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Chalcopyrite	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Quartz	SiO_2	2.65	s. only HF	i.	7
	Calcite	Cu_2S	2.6-2.71	s. HCl	-	3
	Sphalerite	ZnS	3.9-4.1	s. HCl	i.	3.5-4
	Magnetite	Fe_3O_4	5.26	s. a.	i.	5-6
Utah	Sphalerite	ZnS	3.9-4.1	s. HCl	i.	3.5-4
	Quartz	SiO_2	2.65	s. only HF	i.	7
	Chalcedony (opal)	$SiO_2 \cdot nH_2O$	1.73-2.16	-	-	~ 6
	Calcite	$CaCO_3$	2.6-2.71	s. HCl	-	3
	Montmorillonoid	$(Ca \cdot Na)Al \cdot Mg \cdot Fe \cdot Si \cdot Al \cdot Au \cdot Ag \cdot O \cdot H_2O$	2-3	s. s. a.	i.	1-2
	Chlorite	$(MgAl \cdot Fe)_{12} \cdot (SiAl) 8O_{20} \cdot OH_{16}$	-2.6-3.3	sl. s. a.	i.	2-3
	Wollastonite	$CaO \cdot SiO_2$	2.8-2.9	dec. HCl	i.	4.5-5
	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Chalcopyrite	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4

TABLE C-1 (Continued)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
Utah	Galena	PbS	7.57-7.59	dec. HNO_3	-	2.5-2.75
	Bornite	Cu_5FeS_7	5.06-5.08	s. HNO_3	sl. s.	3
	Cubanite	$CuFe_2S_3$	4.50	s. a.	i.	3-3.5
	Molybdenite	MoS_2	4.62-4.73	s. a.	i.	1-1.5
	Arsenopyrite	FeA_5S	5.9-6.2	s. a.	sl. s.	5.5-6
	Stannite	Cu_2FeSn_4	4.3-4.5	s. a.	i.	4
	Garnet	$Ca \cdot Fe \cdot Al \cdot Mg(SiO_4) \cdot X$	3.5-4.25	i.	i.	6.5-7
	Tremolite	$Ca_2Mg_5Si_8O_{22}$-$(OHiF)_2$	3.0	i.	i.	5-6
	Talc	$3MgO \cdot 4SiO_2 \cdot H_2O$	2.6-2.8	s. a.	i.	1
	Serpentine	$3MgO_2(SiO_2) \cdot 2H_2O$	2.5-2.65	dec. a.	i.	2.5-3.5
	Epidote	$Ca_2 \cdot FeAl_2Si_3O_{12}$-$(OH)$	3.4-3.5	i. a.	i.	6
	Gypsum	$CaSO_4 \cdot 2H_2O$	2.32	s. HCl	i.	1.5-2
	Tourmaline	$Al \cdot B \cdot SiO_2 \cdot Li \cdot Fe \cdot Na \cdot Mg$	2.9-3.2	i.	i.	7-7.5
	Graphite	C	2.09-2.23	i.	i.	1-2
	Phodonite	$(MnFeCa)SiO_3$	3.57-3.76	i.	i.	5.5-6.5
	Apatite	$(CaF)Ca_4(PO_4)_3$	3.17-3.23	s. HCl and HNO_3	i.	4.5-5
	Copper	Cu	8.95	s. HNO_3	i.	2.5-3
Michigan	Copper	Cu	8.95	s. HNO_3	i.	2.5-3
	Silver	Ag	10.5	s. a.	i.	2.5-3
	Chalcocite	Cu_2S	5.5-5.8	s. HNO_3	-	2.5-3
	Bornite	Cu_5FeS_7	5.06-5.08	s. HNO_3	sl. s.	3
	Chalcopyrite	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Covellite	CuO	4.6	s. a.	i.	1.5-2
	Digenite	Cu_9S_5	5.5	s. a.	i.	2.5-3
	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Greenockite	CdS	4.5-5	s. HCl	i.	3-3.5
	Sphalerite	ZnS	3.9-4.1	s. HCl	i.	3.5-4
	Galena	PbS	7.57-7.59	dec. HNO_3	-	2.5-2.75
	Calcite	$CaCO_3$	2.6-2.71	s. HCl	i.	3
	Quartz	SiO_2	2.65	s. HF	i.	7
	Dolomite	$CaMg(CO_3)_2$	2.86	-	-	3.5-4
	Barite	$BaSO_4$	4.50	s. a.	i.	3-3.5
	Chlorite	$(MgAlFe)12(SiAl)$-$8 \cdot O_{20} \cdot OH_{16}$	2.6-3.3 -	sl. s. a.	i.	2-3
	Montmorillonite	$(Mg,Ca)O \cdot Al_2O_3 \cdot 5SiO_2 \cdot 2H_2O$	2-3	sl. s. a.	i.	1-2
	Muscovite	$H_2 \cdot KAl_3(SiO_4)_3$	2.76-3	i.	i.	2-2.25
	Orthoclase	$KAlSi_3O_8$	2.55-2.63	i.	i.	6-6.5
	Epidote	$H_2K(Mg,Fe)_3(Al,Fe)$-$(SiO_4)_3$	3.4-3.5	i.	i.	6
	Hematite	Fe_2O_3	5.26	s. a.	i.	5-6
Montana	Acanthite	Ag_2S	7.2-7.3	s. HNO_3	i.	2-2.5
	Aikinite	$PbCuBiS_3$	6.1-6.8	dec. HNO_3	i.	2-2.5
	Alabandite	MnS	4.1	s. HCl	i.	3.5-4
	Alunite	$KAl_3(OH)_6(SO_4)_2$	2.6-2.75	s. H_2SO_4	i.	3.5-4
	Andorite	$PbAgSb_3S_6$	5.33-5.37	sl. s. a.	i.	3-3.5
	Ankerite	$Ca(Fe,Mg)\ (CO_3)_2$	2.9-3.2	s. a.	sl. s.	3.5-4
	Apatite	$Ca_5(F \cdot ClOH)(PO_4)_3$	3.17-3.23	s. a.	i.	4.5-5
	Arsenopyrite	$FeAsS$	5.9-6.2	s. a.	sl. s.	5.5-6.0
	Barite	$BaSo_4$	4.50	s. a.	i.	3-3.5
	Anhydrite	$CaSo_4$	2.8-3.0	s. HCl	sl. s.	3-3.5

TABLE C-1 (Concluded)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
Montana	Bornite	Cu_5FeS_4	5.06-5.08	s. HNO_3	sl. s.	3
	Calcite	$CaCO_3$	2.6-2.71	s. HCl	-	3
	Chabazite	$CaAl_2Si_4O_{12} \cdot 6H_2O$	2.1	d. HCl	i.	4.5
	Chalcocite	Cu_2S	5.5-5.8	s. HNO_3	-	2.5-3
	Chalcopyrite	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Covellite	CuS	4.6	s. HNO_3	-	1.5-2
	Digenite	Cu_9S_5	5.5	s. a.	i.	2.5-3
	Dolomite	$CaMg(CO_3)_2$	2.86			3.5-4
	Enargite	Cu_3AsS_4	4.4-4.5	s. HNO_3	-	3
	Ferberite	$FeNO_4$	7.51	d. a.	i.	4-4.5
	Fluorite	CaF_2	3.18	s. a.	sl. s.	4
	Galena	PbS	7.57-7.59	dec. HNO_3	-	2.5-2.75
	Greenockite	CdS	4.5-5	s. HCl	i.	3-3.5
	Gypsum	$CaSO_4 \cdot 2H_2O$	2.32	s. HCl	i.	1.5-2
	Helvite	$Mn_4SSi_3Be_3O_{12}$	3.2-3.4	s. HCl	i.	6
	Hematite	Fe_2O_3	5.26	s. a.	i.	5-6
	Heulandite	$CaAl_2Si_7O_{18} \cdot 6H_2O$	2.1-2.2	d. HCl	sl. s.	3.5-4
	Huebnerite	$MnWO_4$	7.12	d. HCl	i.	4-4.5
	Magnetite	Fe_3O_4	5.16	s. HCl	-	5.5-6.5
	Marcasite	FeS_2	4.9	s. con. HNO_3	i.	6-6.5
	Molybdenite	MoS_2	4.62-4.73	s. a.	i.	1-1.5
	Orthoclase	$KAlSi_3O_8$	2.55-2.63	i.	i.	6-6.5
	Polybasite	$(Ag,Cu)_{16}Sb_2S_{11}$	6-6.2	dec. HNO_3	i.	2-3
	Proustite	Ag_3AsS_3	5.57	s. HNO_3	i.	2-2.5
	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Quartz	SiO_2	2.65	s. only HF	i.	7
	Rhodochrosite	$MnCO_3$	3.7	s. h. HCl	i.	3.5-4
	Rhodonite	$(Mn,Fe,Ca)SiO_3$	3.57-3.76	sl. s. HCl	i.	5.5-6.5
	Rutile	TiO_2	4.23-5.5	i.	i.	6-6.5
	Scheelite	$Ca(W,Mo)O_4$	6.1	d. HCl	i.	4.5-5
	Siderite	$FeCO_3$	3.96	s. h. HCl	i.	4-4.5
	Sphalerite	ZnS	3.9-4.1	s. HCl	i.	3.5-4
	Stephanite	Ag_5SbS_4	6.2-6.3	s. h. HNO_3	i.	2-2.5
	Stilbite	$CaAl_2Si_7O_{18} \cdot 7H_2O$	2-2.2	dec. HCl	sl. s.	3.5-4
	Stromeyerite	CuAgS	6.1-6.3	s. HNO_3	i.	2.5-3
	Tennantite	$Cu_{12}As_4S_{13}$	4.37-4.49	d. HNO_3	-	3-4
	Tetrahedrite	$Cu_{12}Sb_4S_{13}$	4.4-5.1	d. HNO_3	-	3-4
	Uraninite	$UO_2 \cdot$Th,Zr,La,Y,-Pb,Hc	9-9.7	s. a.	sl. s.	5.5
	Wavellite	$Al_3(OH)_3(PO_4)_2 \cdot 5H_2O$	2.32	s. HCl	s.	3.5-4
	Wolframite	$MnWO_4$	7-7.5	dec. a.	i.	5-5.5
	Wurtzite	ZnS	3.98	s. a.	i.	3.5-4

Pyrite is the characteristic sulfide of the Nonsuch Shale. The principal sulfide mineral of the cupriferous zone at the base of the formation is chalcocite. The succession of minerals outward and upward from the center of the cupriferous zone is native copper, chalcocite, bornite, chalcopyrite, and pyrite. The bornite and chalcopyrite are confined to a narrow fringe at the margin of the cupriferous zone. This pattern of mineral zones is believed to represent reaction between sygentic pyrite and introduced copper.

Butte District, Montana

Butte, Montana, is one of the world's outstanding examples of metal and mineral zoning. Three crudely concentric zones were delineated by Reno Sales in 1913 (2. p. 58): (1) A Central Zone, "occupying an area of altered granite in which the ores are characteristically free from sphalerite and manganese minerals"; (2) An Intermediate Zone, "in which the ores are predominantly copper, but are seldom free from sphalerite"; (3) A Peripheral Zone, "in which copper has not been found in commercial quantities."[2/]

The copper zone can be associated with both the Intermediate and Central Zones. Chalcopyrite is an important mineral at the outer edge of the copper zone in every part of the district. However, deeper within the zone, the arrangement of copper minerals is not smoothly symmetrical with respect to the margin.

The pattern of distribution of Cu-Fe-S series minerals in the large veins of the copper zone is not symmetrical with respect to the metal zones. There is a fringe of chalcopyrite-bornite all around the copper front. However, within the copper zone, pyrite-chalcocite-enargite-covellite-digenite assemblages are prevalent near its eastern edge from the 1,158-meter level to the surface, whereas deep in the large Anaconda veins in the western part of the copper zone, chalcopyrite is dominant. Here the chalcopyrite grades upward into bornite-chalcocite ores and eastward into the high-sulfur assemblages of the Leonard mine.[2/]

Second, copper ores are mined in the Berkeley pit in the southeastern part of the district. At the present time, they account for about half of Butte's production of copper.

Nevada

The Yerington, Lyon County, Nevada, ore body is a disseminated porphyry copper deposit. It is overlain with gravel varying from 0 to 61 meters in thickness. The basement rock is granodiorite intruded by a quartz monozite porphyry. While ore is found in both, most of the ore body occurs within the quartz monozite. Chrysocolla is the chief copper mineral. Most of it is well-disseminated, but the remainder occurs in seams and fractures. The zone of secondary enrichment is usually small, only a minor amount of chalcocite having been found below the oxide zone. In the sulfide zone, chalcopyrite is the principal copper mineral.

The Copper Canyon porphyry copper deposit, consisting of the east and west ore bodies is mainly in fractured and altered sedimentary rocks of the Pennsylvanian age, and adjacent to a granodiorite laccolith. In the east ore body, most ore occurs as hypogene sulfides that replace former hematite and calcite bearing zones of the lower member of the Middle Pennsylvanian Battle Formation. In the west ore body sulfide-rich manto deposits occur generally closer to the granodiorite as replacement of the Pennsylvanian Pumpernickel Formation.[3] The hypogene minerals include pyrite, chalcopyrite, pyrrhotite, and marcasite, with lesser concentrations of arsenopyrite, native gold and silver, and traces of sphalerite, molybdenite, and galena.

Arizona

The sedimentary rocks in the Pima mine area in the Tucson District are thought to be of Permian age. The sequence is as follows: a rusty brown quartzite; a thin-bedded, serpentized, dolomitic limestone; a highly altered (clay-garnet) limestone, these overlain by a massive thick-bedded arkosic quartzite which varies from rusty brown to light gray.

The high-grade copper sulfide ore body is a replacement of chalcopyrite in the clay-garnet material. Within these areas of dissemination, some pyrite is present, both as small veinlets and as disseminated material. Scattered pods of zinc sulfides have been found within the ore zone. Some molybdenite occurs as plating along the minor slips within the ore zone. The main ore minerals within the oxidation zone are native copper and chalcocite. Minor amounts of chrysocolla and azurite are present.

The host rock of the massive chalcopyrite ore bodies is an intensely altered limestone consisting of a mixture of kaolin and small garnets, with occasional remnants of calcium carbonate.

Nearly all of the ore production from the Twin Buttes mines came from a northwesterly trending mineral zone extending through the Copper Glance, Copper Queen, and Copper King mines.

The geology of the Twin Buttes region is composed of quartz monzonite porphyry. The porphyry includes limestone, siltstone, and clastic rocks. The clastic rocks are a thick sequence of interbedded arkoses, arkosic-like pyroclastics, meta-siltstones and siliceous tuffs which uniformly overlie the limestones, siltstones, and quartzites on the south.

On the southwest and northeast, porphyry contacts the sedimentary rocks. These sedimentary rocks are irregularly altered and mineralized. The quartzites are weakly mineralized, but the silicated, garnetized and chloritized limestones and siltstones show the strongest copper mineralization. The clastic rocks are bleached and mineralized in an extensive area on the south with better copper grades occurring to the southeast where they are in contact with the porphyry.

Unlike many of the porphyry copper deposits in the area, the Twin Buttes quartz monzonite porphyry contains scattered, overall weak mineralization.

Chalcopyrite is the most abundant and widespread copper sulfide mineral, but the mineral suite includes secondary chalcocite and covellite and minor bornite. Molybdenite occurs in variable but economically important amounts throughout the mineralized zone. Some sphalerite and galena occur locally. Trace amounts of gold, insignificant silver, and widespread pyrite occur, and in places magnetite is abundant. The two iron minerals are found mostly in the mineralized limestone and siltstone beds.

There are large tonnages of oxidized rock of good copper content in the oxide zone, and in places secondary copper sulfides substantially enhance sulfide ore grade.

Copper minerals in the oxide zone consist principally of chrysocolla and tenorite or melaconite. Cuprite and native copper are locally plentiful, and there are some brachantite and minor malachite.

The Magma mine at Superior, Arizona, has produced over 11,790,000 metric tons of ore yielding 6.8×10^8 kilograms of copper. It is a mesothermal deposit, and although the bulk of the ore has come from the Magma vein, most of the recent production has come from replacement deposits in limestone.

The mine lies in a Precambrian and Paleozoic sequence of rocks that includes schist, diabase, quartzite, shale, and limestone; nearby, these rocks are intruded by igneous bodies of Mesozoic age that were probably the source of the ore-bearing fluids. The area is extensively faulted. There are two principal strike faults of large displacement and numerous smaller steep and flat faults that complicate the prediction of ore occurrence.

The principal copper minerals in the vein are chalcopyrite, bornite, enargite, tennantite, chalcocite, and digenite; these minerals vary in abundance according to the district zoning pattern. The upper part of the mine yielded considerable zinc as sphalerite, and the oxidized zone was rich in silver. The chief gangue materials are pyrite, quartz, and hematite. In the limestone replacement bodies, chalcopyrite and bornite are the chief ore minerals.

Ducktown District, Tennessee

The Ducktown ore deposits have eight massive, sulfide ore bodies that occur in highly folded and metamorphosis graywacke, graywacke conglomerate, mica schists, chlorite-garnet schist, and staurolite schist of Precambrian age. The ore deposits are tabular bodies that have been extensively folded, generally conform to the enclosing rocks, but show local complex differences.

The ore deposits are composed principally of the minerals pyrrhotite, pyrite, chalcopyrite, sphalerite, and magnetite. Gangue minerals are quartz, calcite, actinolite, tremolite, hornblends, garnet, and masses of schistose wall rock. Three fault systems are present that have influenced ore control and ore body configuration. Hydrothermal effects are evident in wall rock alteration. Evidence of retrograde metamorphism is extensive in the wall and ore-associated rock. Preliminary isotopic age dates have indicated four possible metamorphic events during the Paleozoic Era.

Ore genesis is considered to be hydrothermal replacement of receptive beds, predominantly highly calcareous zones, quartzitic zones, and brecciated shear zones. Ore deposition is thought to have occurred during the Devonian Period, with later mobilization and recrystallization of the ore and gangue during Middle and Late Paleozoic times.[2/]

Bingham District, Utah

The Bingham mining district, located in the Oquirrh Mountains about 32 kilometers southwest of Salt Lake City, Utah, is well known as the site of the Kennecott Copper Corporation's porphyry copper deposit situated in and near the Utah Copper or Bingham stock.

Two main types of intrusive igneous rock are present in the district. The Last Chance stock and the southern and eastern portions of the Bingham stock are composed of a relatively dark equigranular-to-seriate porphyritic rock which probably averaged quartz monzonite in composition before alteration and has been called "dark porphyry" and "granite." A second body called "granite porphyry" or "light porphyry" intrudes the north side of the "dark porphyry." This intrusive has the typical porphyry texture with an aphanitic quartz-feldspar groundmass as found at other porphyry copper deposits, and it appears to form a center for the mineralization patterns of the district. Extrusives, although present, contain no ore.

The intrusives consist of granite, quartz, monzonite, granodiorite, quartz diorite, granite porphyry, quartz monzonite porphyry, and biotite quartz latite porphyry. The extrusives consist of andesite, andesite porphyry, andesite breccia, agglomerates, ash deposits, and rarely, basalt.[2/]

The mineralogy of the district is comprised of chalcopyrite, bornite, and molybdenite, with low pyrite. The lead-zinc ore bodies are likewise massive coarse-grained sulfides of galena, sphalerite, with varying amounts of pyrite and with some copper, gold, and silver. While the galena itself contains silver, much of the silver is found as argentiferous tetrahedrite-tennantite.

Gangue minerals associated with the ore are calcite, quartz, silica, saponite, montmorillonoid clay, talc, chlorite, wollastonite, diopside, and mica. Calcite and quartz are by far the most common.

Arizona and Adjacent New Mexico

Arizona and western New Mexico contain two-thirds of the leading copper mines in the United States. Production of molybdenite, lead, zinc, and by-product gold and silver is important.

Precambrian ore deposits are largely limited to the Yavapai Series in Central Arizona, and most of the metal production has come from massive sulfide deposits, dominantly pyrite but containing different amounts of chalcopyrite, tennantite, sphalerite, and galena.

Most of the important ore deposits are of Mesozoic and early Tertiary age, dominated by the porphyry copper deposits spatially related to stocks, plugs, sills, or dikes of quartz-bearing porphyritic to granular rocks. Host rocks of these deposits include the associated intrusive rocks, Precambrian schist and granitic rocks, Paleozoic limestone, and Mesozoic arkose and siltstone. Chalcopyrite and pyrite are the dominant hypogene sulfide materials in the porphyry copper deposits, accompanied by minor molybdenite. Supergene sulfide minerals are chalcocite and minor covellite. Supergene oxide-copper minerals are important locally.

Intense hydrothermal alteration accompanied the metallization in all of the porphyry copper deposits. Five types are recognized: (1) propylitic; (2) argillic; (3) potassic; (4) quartz-sericite (without clay); and (5) lime silicate.[2/]

Breccia pipes, veins, and limestone replacements account for some important deposits of Mesozoic and early Tertiary age, and presumably these are genetically related to the various intrusive rocks of this age.

Lead and Zinc Ores

The common lead minerals are either sulfide ores (galena, PbS), or oxide or carbonate ores (anglesite, $PbSO_4$, and cerussite, $PbCO_3$). Galena is the most abundant lead mineral found in deposits that have been exploited in the United States. Galena is commonly associated with zinc, silver, gold and iron minerals. However, in a few districts the ore bodies are characterized by very simple mineralization, with the lead mineral present to the virtual exclusion of other ore minerals. A noteworthy example is southwestern Missouri, where the lead ores are generally composed of galena in an essentially nonmetallic gangue, with relatively little silver, zinc, or copper present. Table C-2 shows the mineralogy of the major lead-zinc districts.

TABLE C-2

SUMMARY OF MINERALOGICAL SYSTEMS
LEAD ZINC ORE AT MINES (1031)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
New York	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Sphalerite	ZnS	3.9-4.1	s. HCl	i.	3.5-4
	Galena	PbS	7.57-7.59	dec. HNO_3	-	2.5-2.75
	Pyrrhotite	$Fe_{11}S_{12}$	4.58-4.63	dec. a.	sl. dec.	3.5-4.5
	Chalcopyrite	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4.0
	Magnetite	Fe_3O_4	5.26	s. a.	i.	5-6
	Hematite	Fe_2O_3	4.9-5.3	s. HCl	i.	5.5-6.5
	Bornite	Cu_5FeS_4	4.9-5.4	s. a.	sl. s.	3
	Willemite	Zn_2SiO_4	3.9-4.1	s. a.	i.	5-5
	Quartz	SiO_2	2.65	s. only HF	i.	7
	Diopside	$CaMgSi_2O_3$	3.22-3.38	i.	i.	5.5-6.5
	Tremolite	$Ca_2Mg_5Si_8O_{22}(OH,F)_2$	3.0	i.	i.	5-6
	Serpentine	$3MgO_2(SiO_2) \cdot 2H_2O$	2.5-2.65	dec. a.	i.	2.5-3.5
	Talc	$Mg_3Si_4O_{10}(OH)_2$	2.58-2.83	i.	i.	1
	Barite	B_2SO_4	4.50	i.	i.	3-3.5
	Anhydrite	$CaSO_4$	2.98	s. HNO_3	v. s.	3.5
	Chlorite	$Na_5Al_3F_{14}$	3.00	sl. s.	i.	3.5-4
	Ilvaite	$H_2O \cdot CaO \cdot 4FeO \cdot Fe_2O_3 \cdot 4SiO_2$	4.0	gel. HCl	i.	5.5-6
	Grossularite (garnet)	$Ca_3Al_2(SiO_4)_3$	3.53	i.	i.	6.5-7
	Anthophyllite	$(Mg_1Fe)_7Si_8O_{22}(OH,F)_2$	2.85-3.57	i.	i.	5.5-6
	Clinozoisite	$Ca_2Al_3Si_{13}O_{12}(OH)$	3.2-3.4	i.	i.	6.5
	Sericite	$PbCO_3$	6.5	s. HNO_3	i.	3-3.5
	Albite	$NaAlSi_3O_8$	2-63	i.	i.	6-6.5
	Dolomite	$CaMg(CO_3)_2$	2.86	s. a.	sl. s.	3.5-4
Missouri	Galena	PbS	7.57-7.59	dec. HNO_3	-	2.5-2.75
	Sphalerite	ZnS	2.5-2.65	dec. a.	i.	2.5-3.5
	Chalcopyrite	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4.0
	Bravoite	$NoFeS_2$	4.62	s. a.	i.	5.5-6
	Pyrite (rare)	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Marcasite	FeS_2	4.85-4.9	s. con. HNO_3	sl. s.	6-6.5
	Bornite (sparce)	Cu_5FeS_4	4.9-5.4	s. a.	sl. s.	3
	Millerite (sparce)	N_1S	5.3-5.6	s. HNO_3	i.	3-3.5
	Dolomite	$CaMg(CO_3)_2$	2.86	s. a.	sl. s.	3.5-4
	Calcite	Cu_2S	2.6-2.71	s. HCl	-	3
	Dickite	$Al_2O_3 \cdot 2SiO_2 \cdot 2H_2O$	2.62	i.	i.	2.5
	Quartz	SiO_2	2.65	s. only HF	i.	7
	Covellite	CuS	4.6	s. HNO_3	i.	1.5-2
	Chalcocite	Cu_2S	5.5-5.8	s. HNO_3	i.	2.5-3
	Malachite	$CuCO_3 \cdot Cu(OH)_2$	4	s. a.	i.	3.5-4
Colorado	Dolomite	$CaMg(CO_3)_2$	2.86	s. a.	sl. s.	3.5-4
	Quartz	SiO_2	2.65	s. only HF	i.	7
	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Sphalerite	ZnS	2.5-2.65	dec. a.	i.	2.5-3.5
	Galena	PbS	7.57-7.59	dec. HNO_3	-	2.5-2.75
	Chalcopyrite	$CuFeS$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Cerussite	$PbCO_3$	6.53-6.57	s. a.	i.	3-3.5
	Cerargyrite	$AgCl$	5.55	s. NH_4OH	i.	2.5
	Embolite	$Ag(Cl \cdot Br)$	5.8	s. NH_4OH	i.	1.5
	Ankerite	$Ca(Fe \cdot Mn \cdot Mg)(CO_3)_2$	2.8-3.1	s. a.	sl. s.	3.5-4
	Argentite	Ag_2S	7.2-7.4	s. HNO_3	i.	2-2.5
	Barite	$BaSO_4$	4.5	i.	i.	3-3.5
	Rhodochrosite	$MnCO_3$	3.7	s.h. HCl	i.	3.5-4
	Pyrargyrite	$Ag_3 \cdot SbS_3$	5.85	dec. HNO_3	i.	2.5
	Chalcocite	Cu_2S	5.5-5.8	s. HNO_3	i.	2.5-3
	Covellite	CuS	4.6	s. HNO_3	i.	1.5-3
	Anglesite	$PbSO_4$	6.37-6.39	s. a.	sl. s.	2.5-3
	Wulfenite	$PbMoO_4$	6.5-7.0	s. a.	sl. s.	2.75-3

TABLE C-2 (Concluded)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
Colorado	Pyromorphite	$3Pb_3 \cdot P_2O_8 \cdot PbCl_2$	6.5-7.1	s. HNO_3	i	3.5-4
	Smithsonite	$ZnCO_3$	4.42-4.44	s. a.	sl. s.	4-4.5
	Calamine	$2ZnO \cdot SiO_2 \cdot H_2O$	3.4-3.5	gel. a	i.	4.5-5
	Chalcanthite	$CuSO_4 5H_2O$	2.2	s. a.	s.	2.2-2.5
Utah	Galena	PbS	7.57-7.59	dec. HNO_3	-	2.5-2.75
	Sphalerite	ZnS	2.5-2.65	dec. a.	i.	2.5-3.5
	Argentite	Ag_2S	7.2-7.4	s. HNO_3	i.	2-2.5
	Tennantite	$5Cu_2S \cdot 2ZnS \cdot 2AsS_3$	4.37-4.49	dec. HNO_3	i.	3-4
	Proustite	Ag_3AsS_3	5.57	-	i.	2-2.5
	Polybasite	$(AgCu)_{16}Sb_2S_{11}$	6.0-6.2	dec. HNO_3	i.	2-3
	Barite	$BaSO_4$	4.5	i.	i.	3-3.5
	Rhodochrosite	$MnCO_3$	3.7	s. h. HCl	i.	3.5-4
	Calcite	Cu_2S	2.6-2.71	s. HCl	-	3
	Quartz	SiO_2	2.65	s. only HF	i.	7
	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Enargite	$Cu3A5S_4$	4.4-4.5	s. aq. reg. and HNO_3	i.	3
	Tetrahedrite	$(Cu_1Fe)_{12}Sb_4S_{13}$	4.6-5.1	dec. HNO_3	i.	3-4.5
	Altaite	$PbTe$	8.15	s. a.	i.	3
	Bismuthinite	Bi_2S_3	6.75-6.81	s. h. HNO_3	i.	2
	Bornite	Cu_5FeS_4	4.9-5.4	s. a.	sl. s.	3
	Chalcopyrite	$CuFeS$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Chalcocite	Cu_2S	5.5-5.8	s. HNO_3	i.	2.5-3
	Jamesonite	$Pb_4FeSb_6S_{14}$	5.63	s. a.	i.	2.5
	Tetradymite	$Bi_2(S_1Te)_3$	7.2-7.6	s. HCl	i.	1.5-2
	Cerussite	$PbCO_3$	6.53-6.57	s. a.	i.	3-3.5
	Anglesite	$PbSO_4$	6.37-6.39	s. a.	sl. s.	2.5-3
	Pyrolusite	$MnO_2 + 2H_2O$	4.73-4.86	s. HCl	i.	2-2.5
	Brochantite	$CuSO_4 \cdot 3Cu(OH)_2$	3.8-3.9	s. HNO_3	i.	3.5-4
	Copiayite	$(Fe_1Mg)Fe_4(SO_4)_6$- $(OH)_2 20H_2O$	2.08-2.17	s. a.	s.	2.5-3
	Halloysite	$Al_2O_3 \cdot 2SiO_2 \cdot H_2O$	2-2.2	i.	i.	1-2
	Epsomite	$MaSO_4 \cdot 7H_2O$	1.75	s. a.	s.	2-2.5
	Lanarkite	$Pb_2O \cdot SO_4$	6.4-6.8	sl. s. dil. a.	i.	2-2.5
	Malachite	$CuCO_3 \cdot Cu(OH)_2$	4	s. a.	i.	3.5-4
	Melanterite	$FeSO_4 \cdot 7H_2O$	1.9	s. a.	s.	2
	Minium	Pb_3O_4	8.9-9.2	s. a.	sl. s.	2.5
	Smithsonite	$ZnCO_3$	4.42-4.44	s. a.	sl. s.	4-4.5
Idaho	Galena	PbS	7.57-7.59	dec. HNO_3	-	2.5-2.75
	Sphalerite	ZnS	2.5-2.65	dec. a.	i.	2.5-3.5
	Tetrahedrite	$(CuF_2)_{12}Sb_4S_{13}$	4.6-5.1	dec. HNO_3	i.	3-4.5
	Chalcopyrite	$CuFeS$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Pyrrhotite	$Fe_{11}S_{12}$	4.58-4.65	dec. a.	sl. dec.	3.5-4.5
	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Arsenopyrite	$FeS_2 \cdot FeAg_2$	5.9-6.2	dec. HNO_3	sl. s.	5.5-6
	Magnetite	Fe_3O_4	5.26	s. a.	i.	5-6
	Hematite	Fe_2O_3				
	Stibnite	Sb_2S_3	4.61-4.65	s. a.	i.	2
	Uraninite	UO_2 + Th, Zr, La, Y, Pb	9.7	s. a.	sl. s.	5-6
	Gersdorffite	NiS_2NiAs_2	5.6-6.2	dec. h. HNO_3	i.	5.5
	Boulangerite	$Pb_5Sb_4S_{11}$	6.0-6.2	s. a.	sl. s.	2.5-3
	Polybasite	$9Ag_2S \cdot Sb_2S_3$	6-6.2	dec. HNO_3	i.	2-3
	Pyrargyrite	$3AgS \cdot Sb_2S_3$	5.8	dec. HNO_3	i.	2.5
	Bournonite	Pb_5CuSbS_3	5.80-5.86	s. a.	sl. s.	2.5-3
	Scheelite	$CaWO_4$	6.0	dec. HCl	i.	4.5-5
	Barite	B_2SO_4	4.5	i.	i.	3-3.5
	Quartz	SiO_2	2.65	s. only HF	i.	7
	Siderite	$FeCO_3$	3.85	s. h. HCl	i.	3.5-4
	Dolomite	$CaMg(CO_3)_2$	2.86	s. a.	sl. s.	3.5-4
	Ankerite	$Ca(Fe,Mg,Mn)$- $(CO_3)_2$	2.8-3.1	s. a.	sl. s.	3.5-4
	Calcite	Cu_2S	2.5-2.71	s. HCl	-	3
	Chlorite	$Na_5Al_3F_{14}$	3.0	sl. s.	i.	3.5-4
	Hornblende	(Ca,Na,K,Al,Fe)- SiO_2	3.02-3.45	i.	i.	5-6
	Biotite	$K(Mg,Fe)_3AlSi_3O_{10}$- $(OH,F)_2$	2.7-3.1	dec. H_2SO_4	i.	2.5-3
	Garnet	$Fe_3Al_2(SiO_4)_3$	4.25	i.	i.	6.5-7

The important types of lead deposits are the following:[1/]

1. Deposits formed in sedimentary rocks without apparent genetic relationship with igneous rock. They occur as tabular replacements of receptive strata, usually in limestone reefs and dolomites. The ores of this type usually contain galena, sphalerite, and pyrite, but seldom gold, silver, or antimony to any appreciable extent.

2. Deposits formed at shallow or intermediate depths genetically associated with igneous rocks, characterized by complex ores and comprising (a) vein deposits, (b) disseminated pyritic deposits of igneous rocks, and (c) silver-lead replacement in limestone.

3. Deposits in beds formed at high temperature and pressure in, or genetically associated with, igneous rocks. The ore minerals are zinc blende, galena, pyrite or pyrrhotite, quartz, calcite, garnet, and rhodonite.

4. Contact-metamorphic deposits containing characteristic minerals found near contacts of limestone with igneous rocks. The ore minerals are galena and its oxidation products in a gangue of calcite, rhodonite, garnet, pyroxene, hornblende, and tremolite.

The average grade of lead ores mined contain between 3.0 and 8.0 percent lead.

Zinc ores are aggregates of minerals. The most common zinc mineral is sphalerite or zinc blende (ZnS), which with its oxidation products smithsonite ($ZnCO_3$) and hemimorphite ($Zn_4Si_2O_7(OH)_2$-H_2O), forms the chief zinc minerals of the world. Zincite (ZnO), willemite (Zn_2SiO_4), and franklinite ($((Zn,Mn)O \cdot Fe_2O_3)$) occur in a unique and very important zinc deposit at Ogdensburg, New Jersey.[1/]

Sphalerite almost always occurs in association with galena, the sulfide of lead. It may also be associated with copper or other base-metal sulfides or occur alone. Next to iron, cadmium is the most common substitutional impurity in the sphalerite lattice; typical cadmium content of zinc concentrates runs about 0.3 percent.

Most zinc ores occur in limestones and dolomites but also occur in sufficient amounts at many mines, including those of Butte, Montana; the Coeur d'Alene, Idaho; and Ogdensburg, New Jersey.

Missouri

The southeast Missouri lead district located about 113 kilometers south of St. Louis, embraces four important subdistricts and several minor ones. The important subdistricts are Mine La Motte, the Old Lead Belt, Indian Creek, and the Viburnum Lead Belt.

In terms of past production, it is one of the world's largest districts, having produced nearly 9,979,000 metric tons of lead. The vast majority of this production was from the Old Lead Belt centered around Bonne Terre and Flat River. However, the new mines in the Viburnum Lead Belt promise to be at least as productive as the older area.

The lead-producing area of Missouri is located on the flank of a somewhat circular-shaped positive structure known as the St. Francis Mountains, the northeastern part of the Ozark dome. The core of this structure is formed by Precambrian igneous rocks, mainly rhyolitic porphyries intruded by granites and diabasic rocks; they are the oldest rocks in the mining region. Many of the erosion-formed igneous knobs are exposed at surface, and others remain buried by Cambrian and Ordovician sedimentary formations.[2]

The primary sulfide minerals include galena, sphalerite, chalcopyrite, siegenite, bravoite, pyrite, and marcasite.

Galena occurs as bedded or sheet deposits along disconformities and bedded planes, in parts as replacements but locally as open space fillings; as disseminated crystals and crystalline aggregates in several types of dolomite and black shales; and as fractured fillings.

Sphalerite, although a minor constituent for the district as a whole, may be abundant locally, and in these areas commonly is associated with gray shaly dolomite and black shale.

Chalcopyrite is abundant in the eastern part of the district and is rare in the western part. The copper mineral occurs as disseminations in nearly all Bonne Terre lithologies and as thin seams along bedding planes. Where it occurs, chalcopyrite usually is associated with galena.

Siegenite and bravoite are present in small amounts, usually as small disseminated crystals.

Pyrite and marcasite are mixed with all the ore minerals in the area.

Leadville District, Colorado

The Leadville District is on the west flank of the Mosquito Range in central Colorado. The ore deposits are in a sequence of dolomites and quartzites, Cambrian through Mississippian in age and about 152 meters thick, which is extensively intruded by porphyry sills, dikes, and plugs of Tertiary age. The sedimentary rocks and sills dip about 15 degrees east and are broken by many faults, which are predominantly of a northwest trend.2/

The ore deposits are principally manto replacement deposits confined to three dolomite units in the stratigraphic sequence. Of these, the Leadville Dolomite is the most productive. Many replacement bodies have vein roots or branch from veins. The veins occupy faults and are productive mainly in the section of quartzites and dolomites and included sills.

The primary ores consist primarily of pyrite, masmatitic sphalerite, and galena, but locally contain chalcopyrite, silver minerals, bismuth minerals, and gold. Principal gangue minerals are manganosiderite and jasperoid.

Coeur d'Alene District, Idaho

The Coeur d'Alene district in the panhandle of Idaho is one of the major lead-zinc-silver producing areas in the world.

Ore occurs in a series of steeply dipping replacement veins of relatively simple mineralogy. Six periods of mineralization, ranging from Precambrian to Tertiary in age, are recognized. The productive galena sphalerite-bearing and tetrahedrite-bearing veins of the main period of mineralization are younger than the Late Cretaceous stocks but are also probably Cretaceous in age.2/

The primary sulfide minerals include galena, sphalerite, tetrahedrite, chalcopyrite, and pyrite. Principal gangue minerals are pyrrhotite, magnetite, and arsenopyrite.

The major producing mines are clustered in two groups. One group lies north of the Osburn fault in a large triangular-shaped arc that extends north and east of the town of Wallace and is called the Mullan-Burke-Ninemile area. This group includes such famous old mines as the Hecla, Hercules, Stanford-Mammoth, Tamarack, Tiger-Poorman, Frisco, Star, and Morning. The other group lies on the south side of the Osburn fault, and extends from near Wallace westward beyond Kellogg nearly to Pine Creek. This group, somewhat more elongate than the Mullan-Burke-Ninemile area, includes such famous mines as the Galena, Sunshine, and Bunker Hill. Numerous other mines and prospects occur in this group, which also include the productive string of mines along the East Fork of Pine Creek. The eastern third of this group is called the Silver Belt of the Coeur d'Alene district because of the silver content of the ore.

Tintic Mining District, Utah

The East Tintic district in central Utah produces gold, silver, copper, lead, and zinc. All of this ore has been produced from blind ore bodies in Paleozoic sedimentary rocks that are concealed beneath hydrothermally altered volcanic rocks and cut by intrusive bodies of Eocene age. The Paleozoic rocks, which range in age from Early Cambrian to Mississippian, form the core and limbs of a large, north-trending asymmetric anticline that is cut by low-angle thrust faults, high-angle transcurrent faults, mineralized fissures and faults.2/

The ore bodies generally may be grouped into two classes: (1) massive replacement bodies that are rich in silver, lead, zinc, and manganese; and (2) fissure veins that are valuable primarily for their content of gold, copper, and silver.

The primary ore minerals of the replacement deposits are argentiferous galena and sphalerite with some of the silver probably present in blebs of argentite and tennantite in the galena and in small amounts of other silver minerals such as proustite, pearceite, and polybasite. Barite, rhodochrosite, manganosiderite, calcite, and quartz are the gangue minerals. Native gold, enargite, and tetrahetrite are the principal ore minerals of the auriferous copper veins in Tintic Quartzite; and various gold and silver tellurides and tetrahedrite are the chief ore minerals in the gold-telluride veins in Tintic Quartzite. Quartz, pyrite, barite, and clay minerals are the main gangue minerals of the vein deposits.

Balmat-Edwards District, New York

The zinc deposits of the Balmat-Edwards Division of the St. Joseph Lead Company in northern New York provide some 10 percent of the domestic zinc produced annually within the United States. These complex ore bodies are contained within marbles of the Precambrian Granville series, in a repetitive sequence of dolomites and silicated units.

The assemblage of ore minerals is simple. Pyrite and sphalerite are abundant, and galena, pyrrhotite, and chalcopyrite are in minor amounts.

Gangue minerals include quartz, diopside, tremolite, serpentine, talc, carbonate, barite, and anhydrite. In the supergene ores, chlorite is abundant, with minor ilvaite and garnet. In most, if not all cases, the nonmetallic minerals found within ores are either reworked fragments of adjacent wall rock or late products of retrograde metamorphism in the surrounding calcsilicate marbles.2/

There is a wide range of mineral association and texture. Macroscopically, most of the ores are massive to disseminated coarsely crystalline aggregates of sphalerite and pyrite in gangue of quartz, carbonate, and diopside. Where inclusions of wall rock are abundant, the sulfide bodies have the aspect of ore breccias. Pyrite occurs with sphalerite either as cubes and pyritohedra in discrete subhedral or anhedral grains up to 10 centimeters, or in massive aggregates almost without sphalerite or gangue. Sphalerite usually is coarse-grained but ranges from 0.01-millimeter grains to subhedral crystals over 8 centimeters in length.2/

Galena, chalcopyrite, and pyrrhotite are most conspicuous as gash veins and disseminated blebs, often with minor sphalerite, in diopside rock at some distance from zinc ore bodies. Occurrences of chalcopyrite and pyrrhotite within ore bodies can be observed only in polished sections. Chalcopyrite appears as fine blebs and chains within sphalerite and pyrrhotite, and pyrrhotite has complex relations with pyrite and sphalerite.

Gangue minerals may be nearly absent in some of the high-grade ores, or may predominate in disseminated or weakly mineralized zones.

Lehigh County, Pennsylvania

The Friedensville zinc mine of the New Jersey Zinc Company is located about four miles south of Bethlehem, Pennsylvania, in the Saucon Valley, an infolded and down-faulted block of Cambro-Ordovician carbonate sediments surrounded by hills of the Precambrian gneiss and Cambrian quartzite except where breached by Saucon Creek at its entrance into the Lehigh River.

The minerals identified to date are dolomite, calcite, chert, rare microcline feldspar, sphalerite, pyrite, rare chalcopyrite, calamine, smithsonite, rare greenockite, sauconite, and carbonaceous materials and sericite in streaks. The dolomite is present as fragments in the breccia, medium to gray in color, and as white veinlets. Calcite occurs as white veinlets as does the quartz. Chert occurs as ellipsoids, generally flattened.

Sphalerite is generally dark gray in color and on a wet face in the mine is difficult to distinguish from the dark gray host. While in general the grain size is medium to coarse, locally it is flint-like and is characterized by conchoidal fracture.

Pyrite is widely distributed in the ore as masses and in subhedral to enhedral crystals; it commonly occurs banded with the sphalerite. Table C-3 shows the distribution of the mineralogy in the major zinc districts.

Mascott-Jefferson City Zinc District, Tennessee

The Mascott-Jefferson City district currently is the largest zinc-producing area in the United States. The producing mines, Mascott, Davis, Young, New Market Zinc, Coy, and the Jefferson City, are 24 to 45 kilometers northeast of Knoxville in the Valley and Ridge physiographic province, an area between the Great Smoky Mountains and the Cumberland escarpment.

The ore is strata bound, occurring essentially within a stratigraphic range of 61 meters in the Lower Kingsport and Upper Longview formations of Lower Ordovician age. In the lower strata of the mineralized section, the ore generally is found in coarsely crystalline clastic dolomite breccia containing silica and chert debris within a sequence of aphanitic limestones. The upper strata of the mineralized section are fine-grained primary dolomites that, in the ore zones, have been fragmented in mosaic patterns by solution collapse with the interfragmental space filled by white gangue dolomite and sphalerite.

TABLE C-3

SUMMARY OF MINERALOGICAL SYSTEMS
ZINC ORE AT MINES (1031)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
Pennsylvania	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Chalcopyrite	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Calamine	$2ZnO \cdot SiO_2 \cdot H_2O$	3.5	gel. a.	i.	4.5-5
	Smithsonite	$Zn\ CO_3$	4.42-4.44	s. a.	sl. s.	4-4.5
	Greenockite	CdS	4-9	s. a.	i.	3-3.5
	Dolomite	$C_2Mg(CO_3)_2$	2.86	s. a.	sl. s.	3.5-4
	Calcite	Cu_2S	2.6-2.71	s. HCl	-	3
	Sphalerite	ZnS	2.5-2.65	dec. a.	i.	2.5-3.5
Tennessee	Dolomite	$CaMg(CO_3)_2 \cdot CO$	2.86	s. a.	sl. s.	3.5-4
	Chalcopyrite (rare)	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Calcite	Cu_2S	2.6-2.71	s. HCl	-	3
	Sphalerite	ZnS	2.5-2.65	dec. a.	i.	2.5-3.5
New Jersey	Pyrite	FeS_2	5.02	s. HNO_3	sl. s.	6-6.5
	Chalcopyrite	$CuFeS_2$	4.1-4.3	dec. HNO_3	sl. s.	3.5-4
	Calamine	$2ZnO \cdot SiO_2 \cdot H_2O$	3.5	gel. a.	i.	4.5-5
	Smithsonite	$Zn\ CO_3$	4.42-4.44	s. a.	sl. s.	4-4.5
	Greenockite	CdS	4-9	s. a.	i.	3-3.5
	Dolomite	$C_2Mg(CO_3)_2$	2.86	s. a.	sl. s.	3.5-4
	Calcite	Cu_2S	2.6-2.71	s. HCl	-	3
	Sphalerite	ZnS	2.5-2.65	dec. a.	i.	2.5-3.5

The mineralogy is unusually simple, sphalerite being the only important sulfide. Pyrite is very scarce. Dolomite gangue is abundant, silica was deposited in quite small amounts, calcite is minor, and fluorite and arite are very rare.

The principal ore-controlling structures are rubble breccia zones that were formed during the post-Lower Ordovician-pre-Middle Ordovician erosion interval by supergene solution as a phase of the development of a regional karst system of underground drainage that extended to depths of at least 244 meters. The major Appalachian orogenic structures are post-ore and have been superimposed on the ore bodies.2/

Uranium and Vanadium Ores

Uranium metal never occurs in its pure form in nature, but is always present in combinations with other elements, forming a uranium-bearing mineral. There are over 100 uranium minerals. Some occur as primary minerals and many others as secondary minerals. Primary minerals are those that have not been altered since their formation by igneous action. Secondary minerals are those that are formed as a result of decomposition of the original primary minerals.

The four known primary uranium ore minerals are:1/

Uraninite - Crystalline uranium oxide.

Pitchblende - Amorphous uranium oxide.

Davidite - Rare earth-iron-titanium-uranium-oxide.

Coffinite - Adamantine uranium oxide.

The six known secondary uranium ore minerals are:1/

Carnotite - Potassium-uranium vanadate.

Tyuyamunite - Calcium-uranium vanadate.

Torbernite and meta-torbernite - Hydrous copper-uranium phosphates.

Autunite and meta-autunite - Hydrous calcium-uranium phosphates.

Uranophane - Hydrous calcium-uranium silicate.

Schroeckingerite - Complex hydrated sulfate, carbonate and fluoride of calcium, sodium, and uranium.

Vanadium occurs in igneous rocks as well as in sedimentary formations. Alteration of the original mineral is very common, and alteration products are numerous. Patronite (vanadium sulfide), for example, oxidizes to the sulfate; and, in contact with calcium minerals, it is transformed to various hydrated calcium vanadates.

The chief ore minerals of vanadium are patronite, VS_4; carnotite, $K_2O\text{-}2UO_3\cdot V_2O_5\cdot 3H_2O$; roscoelite (vanadium mica); and vanadinite, $Pb(PbCl)(V_2O_4)_3$. Table C-4 shows the mineralogy of the major uranium-vanadium districts.

Grants District, New Mexico

Uranium of the Grants region occurs predominantly in the continental sandstones of the upper part of the Jurassic Morrison Formation, but significant lesser deposits are found in limestone of the Jurassic Todilts Formation and in black shale of the Cretaceous Dakota Formation.[2] The deposits are disseminations that form runs ranging from a few hundred metric tons to several million metric tons. The ore consists mainly of uraninite, uraniferous carbonaceous material, coffinite, and such secondary oxidized minerals as tyuyamunite, carnotite, and uranophane. In sandstone, the ore runs were localized by mudstone, interstitial carbonaceous material, and primary sand channel trends. In limestone, ore deposition was related to folds and fractured zones.

The great bulk of production has come from the Ambrosia Lake mines in the Grants district and the Jackpile and Paguate mines in the Laguna district. The ores, which generally have been 0.20 to 0.30 percent U_3O_8 are treated at mills operated by Kerr-McGee Corporation, United Nuclear-Homestake Partners, and the Anaconda Company, all in the Grants district.

TABLE C-4

SUMMARY OF MINERALOGICAL SYSTEMS
URANIUM-VANADIUM ORE AT MINES (SIC-1094)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
New Mexico	Coffinite	$U(SiO_4)(OH)_4$	5.1	s. a.	i.	5-7
	Uraninite	UO_2	10.96	s·HNO_3 conc. H_2SO_9	i.	5-6
	Tyuyamunite	$Ca(UO_2)_2(VO_4)_2 \cdot 7 \cdot 10 \cdot 5H_2O$	3.7-4.3	s. a.	i.	Soft
	Carnotite	$K_2(UO_2)_2(VO_4)_2 \cdot H_2O$	3.8-4.2	s. a.	i.	1-2
	Zippetite	$2UO_3 \cdot SO_3 \cdot 5H_2O$	3.5	-	-	2-3
	Andersonite	$Na_2Ca(UO_2)(CO_3)_3 \cdot 6H_2)$	2.8-2.86	s. a.	s.	-
	Bayleyite	$Mg(UO_2)(CO_2)_2 \cdot 18H_2O$	2.05-2.06	s. a.	s.	-
	Pyrite	FeS_2	5.02	s. a.	sl. s.	6-6.5
	Barite					
	Calcite	$CaCO_3$	2.6-2.71	s. HCl	i.	3
	Metacorbernite	$Cu(UO_2)_2(PO_4)_2 \cdot 4H_2O_4$	3.22	s. HNO_3	i.	2-2.5
	Phosphuranylite	$Ca(UO_2)_4(PO_4)_2\text{-}(OH_4) \cdot 7H_2O$	3.0	s. a.	i.	2
	Schoepite	$4UO_3 \cdot 9H_2O$	4.8	s. a.	sl. s.	2-3
Wyoming	Uraninite	UO_2	10.96	conc. H_2SO_4 s·HNO_3	i.	5-6
	Pyrite	FeS_2	5.02	s. a.	sl. s.	6-6.5
	Hematite	Fe_2O_3	4.9-5.3	s. HCl	i.	6
	Calcite	$CaCO_3$	2.6-2.71	s. HCl	i.	3
	Coffinite	$U(SiO_4)(OH)_4$	5.1	s. a.	i.	5-7
	Molybdenum	MoS_2	4.7	dec. HNO_3	i.	1-1.5

TABLE C-4 (Concluded)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
Colorado	Uraninite	UO_2	10.96	conc. H_2SO_4 s. HNO_3	i.	5-6
	Coffinite	$U(SiO_4)(OH)_4$	5.1	s. a.	i.	5-7
	Carnotite	$K_2(UO_2)_2(VO_4)_2 \cdot H_2O$	3.8-4.2	s. a.	i.	1-2
	Tyuyamunite	$Ca(UO_2)_2(VO_4)_2 \cdot 7\text{-}10 \cdot 5H_2O$	3.7-4.3	s. a.	i.	Soft
	Rauvite	$CaO \cdot 2UO_3 \cdot 6V_2O_5\text{-}20H_2O$	3.4-3.6	-	-	Soft
	Pyrite	FeS_2	5.02	s. a.	sl. s.	6-6.5
	Uranophane	$Ca(UO_2)_2(SiO_3)_2\text{-}(OH_2)5H_2O$	3.81-3.90	gel. HCl	i.	2-3
	Vanadinite	$Pb_5(VO_4)_3Cl$	6.6-7.1	v.s. HCl	i.	2.5-3
	Pyroxene	$CaMg(SiO_3)_2$	3.2-3.38	i.	i.	5-6
Utah	Uraninite	UO_2	10.96	conc. H_2SO_4 s. HNO_3	i.	5-6
	Coffinite	$U(SiO_4)(OH)_4$	5.1	s. a.	i.	5-7
	Quartz	SiO_2	2.7	s. only HF	i.	7
	Barite	$BaSO_4$	4.3-4.6	i.	i.	2.5-3.5
	Galena	PbS	7.4-7.7	dec. HNO_3	i.	2.5-2.7
	Pyrite	FeS_2	5.0	i.	i.	6-6.5
	Yttrium (rare)	Y	4.34	v.s. dil. s.	sl. dec.	
	Carnotite	$K_2(UO_2)_2(VO_4)_2 \cdot H_2O$	3.8-4.2	s. a.	i.	1-2
	Tyuyamunite	$Ca(UO_2)_2(VO_4)_2 \cdot 7\text{-}10 \cdot 5H_2O$	3.7-4.3	s. a.	i.	Soft

Colorado

The Colorado Plateau region has been the principal domestic source of uranium and vanadium. Most of the deposits occur in streamlaid lenses of sandstone in the Chinle and Morrison Formations. The ore minerals below the zone of oxidation are low-valent oxides and silicates of uranium and vanadium and uraninite, coffinite, montroseite, and vanadium-bearing mica, chlorite, and clay. All of these, except the quite stable vanadium silicates, oxidize readily to a variety of high-valent uranium and vanadium minerals. The common copper sulfides are widespread in sparse amounts and abundant enough in a few places to be ore minerals. Accessory minerals are mainly sulfides; pyrite and marcasite are common but not abundant, and trace amounts of galena and sphalerite are widespread. Minerals containing molybdenum, selenium, chromium, nickel, cobalt, and silver are also present, but only in a few places are these minerals abundant enough to be recognized.

Ores containing uranium and vanadium minerals have been mined from the Salt-Wash Member of the Morrison Formation from many localities in the Colorado Plateau region since about 1900. The most productive deposits are in a relatively small area in southwestern Colorado referred to as the Uravan mineral belt. Principal metals recovered from the Uravan mineral belt ores are uranium and vanadium. Radium was recovered early in the 1900's but no substantial amounts have been recovered since about 1923. Trace amounts of other minerals have been detected in the ores, but none of these occurs in quantities sufficient to warrant recovery. Metals detected other than those found in common rock-forming minerals include molybdenum, copper, silver, selenium, chromium, nickel, cobalt, rare earths, and manganese.

Principal uranium minerals in the unoxidized ores are uraninite and coffinite, while the main uranium mineral in the oxidized ores is carnotite with minor amounts of tyuyamunite. Vanadium-bearing clays, consisting of chlorite and hydromica, are the main vanadium minerals in both the oxidized and unoxidized ores. Montroseite, a low-valent vanadium oxide occurs in significant amounts in the unoxidized ores. The vanadium in the vanadiferous clays is firmly mixed and is relatively unaffected during oxidation; however, montroseite oxidizes readily first to an intermediate mineral, corvusite. Upon further oxidation, the vanadium forms a series of vanadates consisting of carnotite, hewettite, metahewettite, pascoite, rauvite, fervanite, and hummerite.

The uranium and vanadium minerals almost always occur intimately mixed together regardless of the environment containing the mineralization or type of mineral association. Occurrences of one metal without the other in large amounts are virtually unknown in the Uravan mineral belt area. Generally, the amount of vanadium exceeds the uranium in ratios ranging from 3:1 to 10:1 for large quantities.

The intensity of uranium-vanadium mineralization varies considerably throughout a typical deposit, ranging from weakly mineralized rock to ore containing several percent U_3O_8 and V_2O_5. Average shipping grades of typical mines are 0.2 to 0.3 percent U_3O_8 and 1 to 2 percent V_2O_5.

Shirley Basin District, Wyoming

The Wind River Formation of Eocene age is the host rock for large high-grade uranium deposits in the Shirley Basin. The major deposits are in a northwest-trending belt of sandstones that were deposited in stream channels and that were lithologically favorable for uranium accumulation.

The major deposits within the belt are at, or near, the margins of large tongues of altered sandstone formed in the most massive parts of thick sandstone beds. At least two tongues of altered sandstone are present in the belt, one of which is 8 kilometers long, 5 kilometers wide, and 21 meters thick; the other tongue is somewhat smaller.

Ore bodies consist of a few hundred metric tons to several hundred thousand tons of material containing from 0.10 to about 2.00 percent U_3O_8. The ore mineral is uraninite associated with pyrite, marcasite, hematite and calcite.

Mercury Ore

Mercury ore is found in rocks of all geologic ages and all classes. The common host rocks are limestone, calcareous shales, sandstone, serpentine, chert, andestie, basalt, and rhyolite. Two general types of cinnabar ores can be distinguished: (1) disseminated ore, in which the cinnabar has impregnated a more or less fine-grained or highly brecciated gangue; and (2) ore deposited in fissures and cracks in the country rock. Mercury is recovered almost entirely from the sulfide mineral cinnabar (HgS--86.2 percent mercury and 13.8 percent sulfur). The distribution of minerals and the mineralogy of the California mercury district is shown in Table C-5.

TABLE C-5

SUMMARY OF MINERALOGICAL SYSTEMS MERCURY ORE AT MINES (SIC-1092)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
California	Stibnite	Sb_2S_3	4.61-4.65	s. a.	i.	2
	Cinnabar	HgS	8.1	i.	i.	2-2.5
	Pyrite	FeS_2	5.02	s. a.	sl. s.	6-6.5
	Marcasite	FeS_2	4.85-4.9	s. c. HNO_3	sl. s.	6-6.5
	Dolomite (sparse)	$CaMg(CO_3)_2$	2.86	s. h. HCl	sl. s.	3.5-4
	Calcite (sparse)	$CaCO_3$	2.7-2.9	s. HCl	s.	3
	Alunite	$K_2Al_6(OH)_{12}(SO_4)_4$	2.6-2.8	s. H_2SO_4	sl. s.	3.5-4
	Kaolinite	$2(Al_2O_3)\cdot SiO_2\cdot H_2O$	2.59	sl. dec. HCl	i.	2.5
	Halloysite	$Al_2O_3\cdot 2SiO_2\cdot H_2O$	2.0-2.2	i.	i.	1-2
	Montmorillonite	$(MgC_2)O\cdot Al_2O_3\cdot 5SiO_2\cdot 2H_2O$	2.0-3.0	i.	i.	1-2
	Zeolite	$Na_2O.Al_2O_3\cdot 3SiO_2\cdot 2H_2O$	2.2-2.35	gel. a.	i.	5-5.5
	Serpentine	$Mg_3Si_2O_5(OH)_4$	2.55	s. a.	i.	2.5-3.5
	Quartz	SiO_2	~2.65	s. only HF	-	7
	Aragonite	$CaCO_3$	2.93	s. HCl	sl. s.	3.5-4

Sulphur Bank is unique among mercury deposits of the western United States. A major ore body was formed in the recent past by hydrothermal processes that are represented today by actively flowing hot springs. Metacinnabar and stibnite have deposited recently, and it seems probable that some deposition of these minerals still is going on.

The major sulfide minerals are pyrite, marcasite, and cinnabar. Metacinnabar occurs in local concentrations. Stibnite is present in minor amounts.

Miscellaneous Ores

Pure titanium does not exist in natural form because of its great radioactivity with other elements. Minerals rich in titanium are few, and only ilmenite ($FeTiO_3$) and rutile (TiO_2) have commercial importance. Theoretically, pure ilmenite contains 52.67 percent TiO_2, and ilmenite from rock deposits and some sand deposits commonly contains 42 to 50 percent TiO_2. Some sand deposits, however, yield altered ilmenite containing 60 percent or more TiO_2. In rock formations, chiefly anorthosite and related rocks, ilmenite is characteristically associated with iron ores, principally magnetite, but in some places with hematite.

In sand or sedimentary deposits ilmenite occurs with rutile, monazite, zircon, garnet, staurolite, and other minerals of high specific gravity--the so-called "heavy metals." Titanium minerals found in beach sands come from weathered igneous rocks. Fragments of the rocks are carried to secondary deposits (beach, dune, and placer) by erosion or rainfall where wave and wind action aid in concentrating the heavy minerals.

Zirconium is associated with other metals in many minerals, but is recovered only from the minerals zircon and baddeleyite.

Zircon, a zirconium orthosilicate ($ZrO_2 \cdot SiO_2$), grades from colorless to pale yellow, to brownish yellow, to grayish green or reddish brown. It has a hardness of 7.5, a specific gravity of 4.2 to 4.86, a conchoidal fracture, and an adamantive luster.

Baddeleyite, the zirconium dioxide, grades from colorless, to yellow, to brown, to black. It has a hardness of 6.5, a specific gravity of 5.5 to 6.0, a conchoidal to uneven fracture, and a greasy to vitreous luster.

Beryllium

Beryllium is a constituent of over 30 known minerals. The only present commercial source of beryllium, however, is the mineral beryl, a beryllium-aluminum silicate that theoretically can contain 14 percent beryllium oxide. Beryl occurs principally in long, prismatic, hexagonal crystals that usually have a vitreous luster and are emerald green or pale green, grading into light blue, yellow and white. It has a hardness of 7.5 to 8.0, a specific gravity of 2.63 to 2.80. Table C-6 contains the mineralogy for the major deposit in Utah.

Beryl occurs in pegmatites and granites. In pegmatities beryl, if present, is usually disseminated in small grains; occasionally, however, large crystals 0.6 to 0.9 meters in diameter and many meters long are found. Almost all domestic beryl is recovered as a coproduct from the following minerals: feldspar, mica, columbite, tantalite, lithium minerals or cassiterite.

In late 1959, a high concentration of beryllium was discovered in tuff (associated with small amounts of fluoride) in the Thomas Range, Utah. The beryllium mineral was identified as bertrandite ($2Be_2SiO_4 \cdot H_2O$) and occurs on the flats surrounding Spor Mountain in the western part of the Thomas Range in western Juab County, 74 kilometers northwest of Delta, Utah. On the east side of Spor Mountain, these deposits are in quartzanidine crystal tuff of the older volcanic group; on the west side, they are found in the vitric tuff, which commonly contains dolomite pebbles, of the younger volcanic group. The tuff is believed to be favored as the host rock because of its high porosity and permeability. In addition, small beryllium-bearing veins have been found in the Paleozoic dolomite in a few places adjacent to volcanic rocks.

Rare Earth

The rare earths are a closely related family of 15 metals (lanthanum, cerium, praseodymium, neodymium, promethium, samarium, europium, gadolinium, terbium, dysprosium, holmium, erbium, thulium, ytterbium, and lutetium). The principal commercial source of these metals has been the mineral monazite, a rare earth phosphate containing thorium; but the discovery in 1949 in California of a large deposit of bastnaesite, a fluorocarbonate of the rare earths, added another source to the United States supply. The rare earth silicate minerals, cerite and allanite, are considered commercial sources.

TABLE C-6

SUMMARY OF MINERALOGICAL SYSTEMS BERYLLIUM ORE AT MINES (SIC-1099)

State	Mineral	Chemical Composition	Specific Gravity	Solubility Reagents	Solubility Water	Hardness
Utah	Bertrandite	$Be_4SiO_7(OH)_2$	2.6	i.	i.	6
	Quartz	SiO_2	2.65	s. HF	i.	7
	Chlorite	$12(Mg\cdot Al\cdot Fe)\cdot(SiAl)8\cdot O_{20}(OH)_{16}$	2.6-3.3	i.	i.	2-3
	Biotite	$K(MgFe)\cdot 3\cdot AlSi_3-O_{10}(OHF)_2$	2.7-3.3	dec. H_2SO_4	i.	2.5-3
	Hornblende	$K\cdot CaAl,Fe,Mg\cdot Na\cdot SiO_2$	3-3.5	i.	i.	5-6
	Augite	$CaMg(SiO_2)_2Mg-AlFeSiO_6$	3.2-3.5	i.	i.	5.5-6
	Hypersthene	$FeMgSiO_3$	3.4-3.5	i.	i.	5-6
	Apatite	$(CaF)Ca_4(PO_4)_3$	3.17-3.23	s. a.	v. sl. s.	4.5-5
	Zircon	$ZrO_2\cdot SiO_2$	4.7	i.	i.	5-6
	Topaz	$[Al(FOH)]_2SiO_4$	4.7	i.	i.	7.5
	Tourmaline	$AlBSiO_3\cdot Li,Fe,Mg$	2.9-3.2			7-7.5
	Rutile	TiO_2	4.26	s. H_2SO_4	i.	6-6.5
	Cassiterite	SnO_2	7.0	s. conc. H_2SO_4	i.	6-7
	Cristobalite	SiO_2	2.27	i.	i.	6-7
	Tridymite	SiO_2	2.28-2.33	s. h. NA_2CO_3	i.	7
	Montmerillonite	$(MgCa)\cdot O\cdot Al_2O_3\cdot 5SiO_2\cdot 2H_2O$	2.0-3.0	i.	i.	1-2
	Calcite	$CaCO_3$	2.7-2.9	s. HCl	s.	3
	Ankerite	$Ca(Fe,Mg,Mn)(CO_3)_2$	2.8-3.1	s. a.	i.	3.5-4
	Opal	$SiO_2\cdot nH_2O$	1.9-2.3	s. HF	i.	5.5-6.5
	Chalcedony	SiO_2	2.6-2.64	i.	i.	7
	Orthoclase	$KAlSi_3O_8$	2.57	i.	i.	6
	Fluorite	CaF_2	3-3.3	s. a.	i.	4
	Limonite	$Fe_2O_3\cdot H_2O$	3.6-4.0	s. HCl	i.	5-5.5
	Dolomite	$MgCA(CO_3)_2$	2.86	s. h. HCl	sl. s.	3.5-4
	Black Manganese Oxide					
	Psilomelane	$MnO_2(B2,K_2,Na\ )-O,H_2O$	3.3-4.7	s. HCl	i.	5-7
	Pyrolusite	$MnO_2\cdot 2H_2O$	4.73-4.86	s. HCl	i.	2-2.5

A massive barite-carbonate-bastnaesite lode is located in the Mountain Pass area of San Bernardino County, California. This deposit of bastnaesite, a fluorocarbonate of the rare earth metals, was discovered in April 1949. The main ore body is about 762 meters long and 152 meters wide. A tremendous tonnage is available, with a grade of about 10 percent total rare earth oxides.[2]

Antimony

Antimony occurs in many ore minerals. The most common ores, with their compositions, are shown in Table C-7.

TABLE C-7

MINERALS OF ANTIMONY[1]

Name of Mineral	Formula	Percent Antimony
Stibnite	Sb_2S_3	71.4
Stibiconite	$H_2Sb_2O_5$	74.5
Senarmonite	Sb_2O_3	83.3
Cervantite	$Sb_2O_3Sb_2O_5$	78.9
Valentinite	Sb_2O_3	83.3
Livingstone	$HgS - 2Sb_2S_3$	53.0
Jamesonite	$Pb_2Sb_2S_5$	29.4

Antimony deposits may be classified into two distinct general types, but gradations exist within the two. The first is the simple type, consisting primarily of antimony minerals in a siliceous or carbonate gangue with only small or negligible quantities of other metals. In almost all deposits, the original antimony mineral is stibnite, the antimony sulfide, but native or metallic antimony is occasionally found. Where the ores have been exposed to extensive oxidation, the original stibnite has been entirely converted to antimony oxides.

The second type of deposit is mineralogically complex and is very commonly also structurally complex. In many places, stibnite is the antimony-bearing mineral, but often the metal is only one component of minerals that also contain lead, copper, silver, and mercury. In addition, in most deposits of the complex type, other nonantimony-bearing metallic minerals are present and are commonly more abundant than the antimony-bearing minerals. With few exceptions, the ores are mined primarily for lead, gold, silver, mercury, zinc, copper, or tungsten. The antimony may be a by-product of only minor value; or, in some instances, the value of the ore decreases because of the antimony content.

The Sunshine Mine in the panhandle of Idaho is the major antimony producing mine in the United States. The mine lies on the south side of the Osburn fault and extends from near Wallace, westward beyond Kellogg to Pine Creek, and is adjacent to the Mullan-Burke-Ninemile area.

Ore occurs in a series of steeply dipping replacement veins of relatively simple mineralogy. Six periods of mineralization, ranging from Precambrian to Tertiary in age, are recognized.

The major vein minerals at the Sunshine Mine are: siderite, quartz, pyrite, tetrahedrite, chalcopyrite, galena, and sphalerite. The valuable minerals are tetrahedrite $(Cu, Fe, Zn, Ag)_{12} Sb_4 S_{13}$, which contains silver, antimony and copper; chalcopyrite ($CuFeS_2$), which contains copper, and galena (PbS), which contains lead. Tetrahedrite is the most important ore mineral. Some of the tetrahedrite occurs as massive sulfide, but most of it occurs as disseminated particles and veinlets in siderite and quartz. The chalcopyrite and galena content are very erratic, varying from 0.01 to 0.2 percent in the mill heads.

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APPENDIX D

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GLOSSARY

Adit - A horizontal passage driven from the surface for the working of a mine.

Anode - Electropositive pole of an electrolytic cell.

Anolyte - The electrolyte adjacent to the anode in an electrolytic cell.

Bench - A ledge which in open-pit mines forms a single level of operation above which ore or waste are excavated from a continuous face.

Blading - Digging and pushing dirt without picking it up.

Berm - A horizontal shelf or ledge built into an embankment of sloping wall of an open pit, to break the continuity of a long slope to increase the stability.

Burn - To pulverize first with very heavy explosive charges.

Burn Cut - A drill hole pattern widely used in fast moving tunnels. Holes are uncharged and serve as a relief zone.

Breasting - A short leading stall worked at right angles to, and forming the face of the main levels.

Breast Stoping - A method of stoping employed on veins where the dip is not sufficient for the broken ore to be removed by gravity.

Borehole - A hole made with a drill, auger or other tools used in mining.

Backfill - Waste sand or rock used to support the roof of a mine after removal of the ore.

Cathode - The electrode where electrons enter (current leaves) an operating system such as an electrolytic cell.

Cementation - The action of precipitating copper from a solution of copper sulfate by addition of iron to the solution.

Coffer Dam - A temporary watertight enclosure from which the water is pumped to expose the bottom of a body of water and permit construction as of foundations or piers.

Concentration - Separation and accumulation of economic minerals from gangue (sometimes called milling).

Concentrator - A plant where ore is separated into concentrates and tails.

Concentrate - The valuable material recovered from ores.

Competent - Rock formations in which no artificial support is needed to maintain a cave free borehole.

Countercurrent Decantation - The clarification of water and the concentration of tailings by the use of several thickeners in series. The water flows in the opposite direction from the solids.

Cyclone Classifier - A device for classification by centrifugal means of fine particles suspended in water, whereby the coarser grains collect at and are discharged from the apex of the vessel, while the finer particles are eliminated with the bulk of the water at the discharge orifice.

Drifter - A person skilled in the use of air-driven percussive rock drills and other processes used in excavating tunnels or passages.

Drift - A horizontal passage underground which follows the vein of ore.

Dike - An embankment of earth or stone used to contain concentrator tailings.

Dump Leaching - Term applied to dissolving and recovering minerals from sub-grade ore materials from a mine dump. The dump is irrigated with water, sometimes acidified, which percolates into and through the dump, and runoff from the bottom of the dump is collected and mineral in solution is recovered by a chemical reaction.

Electrolysis - Chemical change resulting from the passage of an electric current through an electrolyte.

Electrolyte - A nonmetallic electrical conductor in which current is carried by the movement of ions instead of electrons with the liberation of matter at the electrodes.

Froth Flotation - A flotation process in which the minerals floated gather in and on the surface of bubbles of air or gas.

Gangue - Undesired minerals associated with ore, mostly nonmetallic. It is usually rejected as tailings in concentrating ores.

Heap Leaching - A process used for the recovery of copper from weathered ore and material from mine dumps. The material is laid in beds alternately fine and coarse until the thickness is roughly 20 ft. It is treated with water or the spent liquor from a previous operation. Intervals are allowed between watering to allow oxidation to occur. The liquor seeping through the beds is fed to tanks, where it is treated with scrap iron to precipitate the copper from solution.

Hummocky Surface - A surface which is lumpy or has small uneven knolls.

In-situ Leaching - Leaching conducted on an ore body that is not removed from its original or natural position.

Keyway - A structure used in tailings pond dams to provide strength and stability. The keyway is constructed below and above ground surface, and the dam is built over the keyway.

Launder - An open channel or trough in which scrap iron is contained. Pregnant liquor is channeled through the iron to precipitate copper.

Lift - The method used to raise the level of ore, for leaching operations, to build the final heap.

Long Wall Retreating - A mine system of long wall working in which the developing headings are driven narrow to the boundary or limit line, and the ore seam is extracted by long wall faces retreating towards the shaft.

Mesothermal Deposit - A mineral deposit formed at moderate temperature and moderate pressures, in and along fissures or other openings in rocks, by deposition at intermediate depths, chiefly from hydrothermal fluids derived from consolidating intruding rocks.

Muck - Rock or ore broken in the process of mining.

Mucker - A laborer who loads broken ore into cars for transport.

Middlings - Particles incompletely liberated into concentrate or tailings. Generally treated in another process to produce a secondary concentrate.

Overburden - Material of any nature that overlies an ore deposit.

Panel Mining - A system of ore extraction in which the ground is laid off in separate districts or panels with pillars of extra size being left between the panels.

Plutonic Rock - Igneous rocks formed deep within the earth under the influence of heat and pressure, solidified from a molten mass.

Pregnant Solution - Solution containing a recoverable concentration of metals.

Precipitation - The act of separating a solid form from a solution by chemical means.

Proctor Density - A rating given to soils after compaction tests have been made. The result can be correlated with the moisture content and the load bearing ability of the soil.

Raise - A mine opening driven upward from the back of a level to a level above.

Raffinate - The aqueous solution remaining after metal values have been extracted by the solvent.

Refinery - A term sometimes applied to the plant in which metal or valuable mineral is extracted from an ore or concentrate.

Rippers - An accessory that is either mounted or towed at the rear of a tractor and generally used in place of blasting as a means of loosening compacted soils and soft rocks for scraper loading. The ripper has long, angled teeth that are forced into the ground surface, ripping the earth loose.

Scraper - A surface vehicle mounted on large rubber-tired wheels, fitted at the bottom with a cutting blade, and pushed by tractor. As the vehicle moves along, the cutting blade loads a bucket contained in the vehicle. When full the scraper is transported to a dumping point where the material is discharged through the bottom of the vehicle in an even layer.

Skip Hoist - A bucket or car operating up and down a defined path, receiving, elevating, and discharging bulk materials.

Slimes - A material of extremely fine particle size encountered in ore treatment. For example, a product of wet-grinding containing valuable ore in particles so fine, as to be carried in suspension by water.

Smelter - A plant where ore concentrates are melted in a furnace in order to recover the metal values.

Spent Liquor - Liquor that has had all metallic values removed by processing.

Solvent Extraction - A method of separating one or more substances from a mixture by treating a solution of the mixture with a solvent that will dissolve the required substance.

Stabilization - Chemical, mechanical or vegetative treatment designed to increase or maintain the stability of soil and improve its engineering properties.

Stope - An excavation from which ore has been taken in a series of steps.

Stoping - Excavating ore by means of a series of horizontal, vertical, or inclined workings in veins or large irregular bodies of ore, or by rooms in flat deposits.

Stopping - A permanent wall built to close off the unused, or no longer needed crosscuts in a mine.

Tailings - The portions of concentrated ore that are too poor in mineral values to be treated further. Disposed of as a waste material.

Tail Water - Water immediately downstream from a structure.

Thickeners - A vessel or apparatus for reducing the proportion of water in a pulp.

Tuff - A rock formed of compacted volcanic fragments generally smaller than four millimeters in diameter.

Vat Leaching - Leaching of copper ore with sulfuric acid in an open tank.

Waste Rock - Barren or submarginal rock which has been mined, but is not of sufficient value to warrant treatment and is therefore removed ahead of the concentrator.

Wet-grinding - The milling of materials in water or other liquid.

UNITS AND CONVERSION FACTORS

	Multiply by	
cm	2.54	in.
in.	0.3937	cm
ft	0.3048	m
m	3.281	ft
mile	1.609	km
km	0.6214	mile
ft^2	0.0929	m^2
m^2	10.764	ft^2
acre	0.4047	hectare
hectare	2.471	acre
ft^3	0.0283	m^3
m^3	35.3356	ft^2
gal.	3.785	liter
gal.	0.003785	cu m
cu m	264.2	gal.
lb	0.4536	kg
kg	2.2046	lb
oz (troy)	31.103	g
oz (troy)/ton	0.0342	kg/MT
ton	0.907	MT
MT	1.1023	ton
long ton	1.016	MT
MT	0.9842	long ton

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